Atmospheric Acid Leaching of Oxidised and Mixed Copper Cobalt Ores Mined in the Democratic Republic of the Congo

by

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ABSTRACT

Kamoto-Oliveira-Virgule (KOV) located in the Democratic Republic of the Congo, operates its mining operation on the north-western boundary of the approximately 350km long Congolese copper belt. This deposit accounts for nearly half of all the Cu-Co resources within the DRC.

The concentrator and refinery being studied was launched as a brownfields project, designed to receive concentrated copper oxides that feed the leaching circuit directly. Copper sulphide concentrate was received by the fluidized bed roasters after sulphide and oxide flotation.

Current open pit operation has predominantly oxidised copper ore and mining the mineral resources from the open pit is considerably more economical than running the underground operation that produces predominantly copper sulphides. Consequently, the decision to build a Flotation Tailings Acid Leach (FTAL) plant was made which allows for the copper oxide flotation process to be eliminated completely by increasing the leach circuit capacity to process all the ore from the open pit operation, the ore will only undergo a pre-flotation process to recover the copper sulphides fraction.

The justification for the FTAL plant lies with the mineral recovery gained by eliminating the copper oxide flotation circuit. The purpose of this study is to characterize the ideal leaching conditions of flotation tails for ore mined from KOV and its respective orebodies. Therefore, determining if the mining, concentration and refining operation can run a metal recovery process from mine to metal efficiently by evaluating the leach performance and characteristics of all the available resources including the ores containing dolomite and calcite.

Core samples were received from FNSR, Oliveira, Virgule and Variant ore bodies in the KOV pit. The samples were crushed and milled to a desired P_{80} particle size followed by a 4-minute flotation step to remove the copper sulphide minerals present in the sample. The concentrate from this step was submitted to the laboratory for analysis. The flotation tailings were dried and sent for chemical analysis and labelled as the leach feed.

The flotation tails were then re-pulped and vigorously agitated and leached at a controlled temperature using diluted sulphuric acid, at the desired pH for a period of 4 hours. After the leach process, the acidic slurry was filtered, and washed.

The initial experimental results revealed that particle with a P_{80} of 75µm, 150µm and 212µm had leach recoveries of 91%, 89%, 89% respectively and average acid consumption values of 141, 132 and 128 kg/t respectively, but the sulphides fraction recovery dropped by 20% from 212µm to 150µm.

The fresh acid consumption (FAC) decreased from 142kg/MT to 86kg/MT for leach tests performed at pH values of 1.0 and 2.0 respectively and leach recovery decreased from 95.4%$_{CuOx}$ to 93%$_{CuOx}$ respectively. The effect on cobalt however was much more pronounced as the total cobalt recovery dropped from 82% to 60% for pH values of 1.0 and 2.0 respectively. A change in the percentage solids in the leach slurry showed that the optimal leach conditions was at 30% solids with a copper leach recovery of 98.2%.

Leaching at 30°C, 45°C and 60°C resulted in copper leach recoveries of 98.2%, 98.2% and 98.5% respectively. The leaching of cobalt was much more affected by the change in temperature, the total cobalt leach efficiency varied from 78.6% to 88.0% for tests conducted at 30°C and 60°C respectively.

A very strong correlation between the contained calcium in the feed and the gangue acid consumption value was found, which would make it uncomplicated to create an advanced blending strategy if the operation would invest in online analysers placed on the conveyors that feed the stockpiles.

Furthermore, the mineralogy revealed that the percentage cobalt in the ore had a profound precipitation effect on the already leached copper in solution due to the electron negativity of the Co^{3+} found in heterogenite.

The optimised leaching conditions were used to leach a large quantity core samples to verify the results from the initial core samples.

ABSTRAK

Kamoto-Oliveira-Virgule (KOV) in die Demokratiese Republiek van die Kongo, bedryf sy mynbedryf op die Noord-Westelike grens van die ongeveer 350 km lang Kongolese koperbelt. Hierdie neerslag dra by tot bykans helfte van al die Cu-Co hulpbronne binne die DRC verantwoording doen.

Die konsentrator en raffinadery wat bestudeer is, is bekendgestel as 'n beboudprojek ontwerp om gekonsentreerde koperoksiede te ontvang wat die logingskring direk voer, en kopersulfiedkonsentraat is ontvang deur die fluïedbedroosters na sulfied- en oksiedflotasie.

Huidige oopgroefmynbedrywighede het egter hoofsaaklik geoksideerde kopererts en om die mineraalhulpbronne vanuit die oopgroefmyn te myn is aansienlik meer ekonomies as om die ondergrondse operasies te bedryf wat hoofsaaklik kopersulfiedes produseer. Dus is daar besluit om 'n FTAL-aanleg te bou wat toelaat dat die koperoksiedflotasieproses heeltemal geëlimineer word. Deur die logingskringkapasiteit te verhoog om al die erts van die oopgroefmynbedryf te prosesseer, sal die erts slegs 'n voorflotasieproses ondergaan om die kopersulfiedfraksies te herwin.

Die regverdiging vir die FTAL-aanleg lê daarin dat mineraal herwinning verkry word deur die koperoksiedflotasiekring te elimineer. Die doel van hierdie ondersoek is daarom om vas te stel of die myn, konsentraat en affineringbedryf 'n metaalherwinningproses van myn tot metaal doeltreffend kan bedryf deur die loging werkverrigting en kenmerke van al die beskikbare hulpbronne, insluitend die erts wat dolomiet en kalsiet bevat, te evalueer.

Die eksperimentele resultate het gewys dat partikel groottes van P80 van 75 µm, 150 µm en 212 µm loging herwinning van 91%, 89% en 89% onderskeidelik gehad het, en gemiddeld suur gebruik waardes van 141 kg/t, 132 kg/t en 128 kg/t onderskeidelik, maar die sulfiedfraksie herwinning het met 20% geval van 212 µm na 150 µm.

Die vars suur gebruik (FAC) het van 142 kg/MT na 86 kg/MT verminder vir logingstoetse uitgevoer by pH-waardes van 1.0 en 2.0 onderskeidelik en loging

herwinning het van 95.4%CuOx na 93%CuOx onderskeidelik, verminder. Die effek op kobalt was egter baie duideliker omdat die totale kobalt herwinning van 82% tot 60% vir pH-waardes van 1.0 en 2.0 onderskeidelik, geval het. ’n Verandering in die persentasie vastestowwe in die logingslyk het gewys dat die optimale loging toestande by 30% vastestowwe met ’n koper loging herwinning van 99.6% was. Loging by 30 °C, 45 °C en 60 °C het loging herwinning van 99.6%, 97.0% en 98.2% onderskeidelik, tot gevolg gehad.

Dis bewys dat die loging van kobalt baie meer geraak word deur die verandering in temperatuur en pH wat gevarieer het van 78.6% tot 88.0% vir toetse uitgevoer by 30 °C en 60 °C onderskeidelik.

‘n Baie sterk korrelasie was gevind tussen die kalsium inhoud en die afvalers suur gebruik (GAC), dit stel die myn in staat om ‘n eenvoudige erts meng strategie te ontwikkel indien hulle in aan-lyn analiseerders sou belê wat op die vervoerbande wat die voorraad hope over geplaas kan word.

En nog meer, het die mineralogie bekendgemaak dat die persentasie kobalt in die erts a merkwaardige presipitasie effek op die alreeds geloogde koper in oplosing het as gevolg van die electron negetiwiteites verskil van die Co^{3+} in die heterogeniet.

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1 Introduction and background

1.1 Background

The operation under consideration, see Figure 4, is situated in the Democratic Republic of the Congo, the mining operation is located on the north-western boundary of the approximately 350km long Congolese copper belt (Porter GeoConsultancy Pty. Ltd, 2013). This deposit contributes to the largest cluster of the Roan group of Cu-Co resources, accounting for nearly half of all the Cu-Co resources within the DRC.

The first copper produced from the Kamoto-Oliveira-Virgule (KOV) deposit date back to the early 1900s. The operation was launched as a brownfields project, and the refinery was successfully rehabilitated by 2008, by 2014 the production target was ±300 000T of copper cathode annually, but officially only produced 158 026 tones of copper. The mining and refinery operation was placed on care & maintenance in the 4th quarter of 2015, in preparation for constructing a new Flotation Tailings Acid Leach plant (FTAL).

The aim of this investigation is to determine if the company can run a mining operation efficiently from the open pit mining deposits by evaluating the leach performance and characteristics of all the available resources including the ores containing dolomite, calcite as well as other gangue minerals.

1.2 Mining operations, ore body and lithologies

The mining operation being studied is located in the Democratic Republic of the Congo and has two open pit mines namely Kamoto Oliveira Virgule (KOV) and Mashamba East (ME) which can be seen in Figure 1 and Figure 2 respectively. KOV has 5 Ore bodies namely, FNSR, Oliveira, Variante, Virgule and Kamoto East. Mashamba East is a far smaller pit than KOV and is considered a separate open pit operation and has only one ore body namely Mashamba East.

The purpose of this study is to characterize the ideal leaching conditions of flotation tails for ore mined from KOV and its respective orebodies. Brief mention of tests done on ME may be mentioned in the thesis but was not the focus of the study.

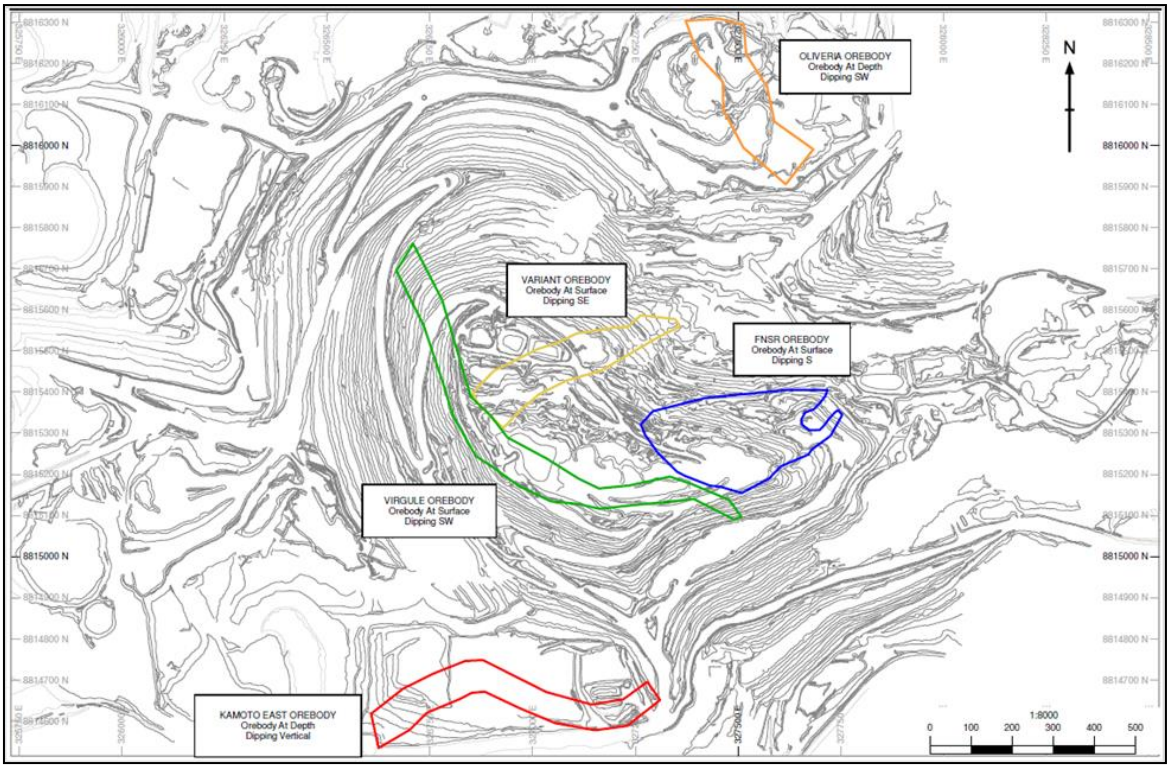


Figure 1: Kamoto Oliveira Virgule open pit mine

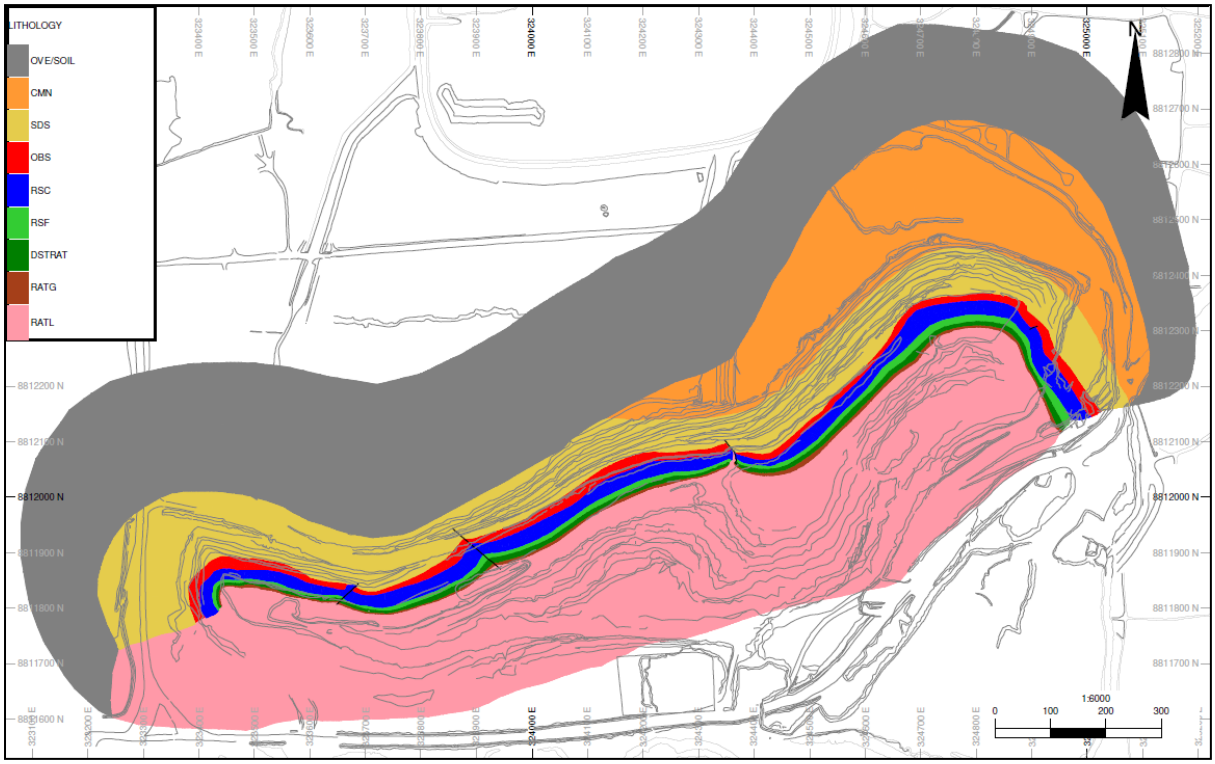


Figure 2: Mashamba East ore body

Each ore body consists of 11 lithologies of which 7 are profitably minable ores and can be seen in Figure 3. All ore with a total copper content less than 0.65%$_{wt}$ is regarded as waste material. The cut-off grade is a function of the copper price, selling cost, processing cost, administration cost, royalties and a copper recovery factor. The cut-off grade has been determined by the business unit's financial department, and is therefore not investigated.

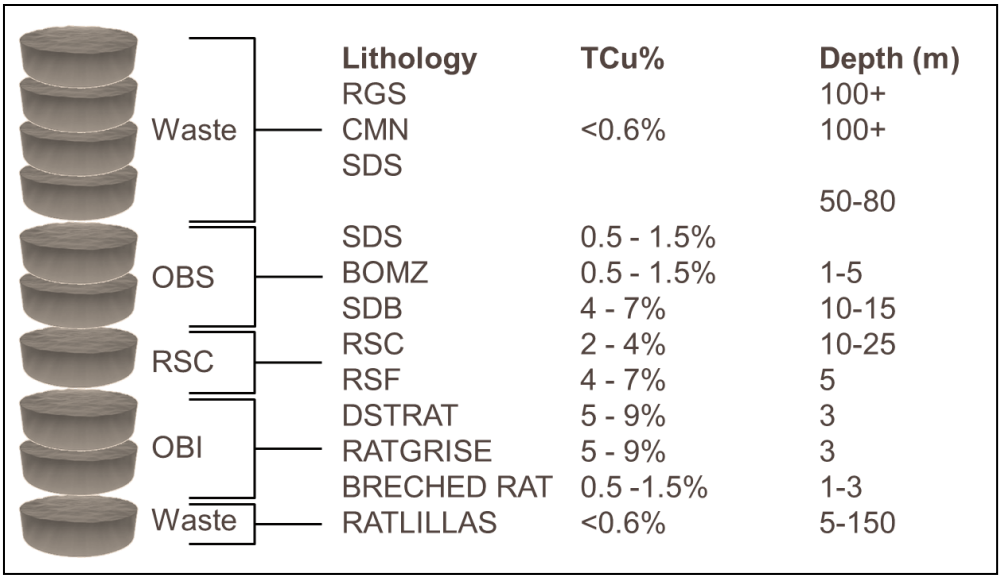


Figure 3: Lithologies and associated head grades

1.3 Mineralogy

The 7 lithologies that is of the interest due to their grade and overall abundance are:

1. SDS - Schistes Dolomitiques Superior
2. SDB - Schistes Dolomitiques Base
3. RSC - Roches Siliceuses Cellulaires
4. RSF - Roches Siliceuses Feuilletees
5. DSTRAT – Dolomitic Stratification
6. RATGRIS - Roches Argilo-Talqueuses Gris
7. RATLILlAS - Roches Argilo-Talqueuses Lilas

Based on the geological classification there are certain minerals that are expected to be abundant in the lithologies, a brief overview of each lithology will now follow.

RATLILLAS, Roches Argilo-Talqueuses Lilas, is characterised by the absence of sulphides and a uniform red colour, because of the presence of disseminated hematite. Quartz, micas and chlorite are abundant in most bands, but there is also a presence of dolomite.

RATGRIS otherwise known as Roches Argilo-Talqueuses Grey (Gris in French) composed of grey, chloritic and dolomitic silt- to sandstone, and is distinguished by its grey colour and the absence of hematite, which is replaced by sulphides (pyrite and chalcocite). Essentially it is un-stratified sandstone made up of predominantly angular quartz (Porter GeoConsultants, 2012).

RSC, Roches Siliceuses Cellulaires, overlies the RSF with a well-defined boundary between the lithologies. The lithology is identified by its cellular siliceous cavities at the weathered top part of the lithology. Below the level of weathering and oxidation, it is a coarsely crystalline, silicified dolomitic unit, composed almost exclusively of dolomite and authigenic quartz. The silicification is more intense along fractures, enclosing less altered dolomite which weathers to leave the honeycomb of silica and rock cavities seen at the surface. This phenomenon causes the RSC to have a lower gangue acid consumption and the presence of supergene malachite in the rock cavities results in RSC being a high-grade ore. Supergene enrichment occurs at the base of the oxidized portion of an ore deposit. Metals that have been leached from the oxidized ore are carried downward by percolating groundwater and react with hypogene sulphides. The interaction between the oxide and sulphide phases produce secondary sulphides with metal contents higher than those of the primary ore.

RSF, Roches Siliceuses Feuilletees, is identified by its grey to light brown colour, and consist of highly silicified dolostones, and typically contains more siliceous minerals that doesn't consume acid as regularly as the calcium carbonates. But due to RSF and DSTRAT being adjacent to one another the contact areas can be difficult to define and individual RSF samples could result in higher acid consumption. Together the RAT, DSTRAT and RSF forms what is known as OBI, Ore Body Inferieure, or lower ore body.

SD, Schistes Dolomitiques, zone comprised of SDS, Schistes Dolomitiques Superior and SDB, Schistes Dolomitiques Base, they are composed of alternating laminated, locally carbonaceous, dolomitic mudstone and siltstone beds. This would indicate that the ore from these lithologies will be high in acid consumers and that the leach efficiency will be lower due to clays that cause surface coverage issues during leaching.

1.4 Historic operation circuit

The historical and proposed process flow diagrams can be seen in Figure 4(a). For ease of explanation, the process changes have been highlighted in the process flow diagrams (PFDs).

The process from mine to metal starts at the open-pit and underground mines which feeds the mills dedicated to the copper oxide material mills and copper sulphide material mills respectively.

The milled copper sulphide material was submitted to a conventional flotation process. Copper sulphides are naturally hydrophobic and flotation recoveries of more than 80% are easily achievable, as opposed to copper oxides which are hydrophilic. A copper concentrate of between 34% and 38% was produced. The concentrate then reported to a roaster. During the roast at ±686°C, the process of sulphation, creates a copper product known as calcine that is readily dissolvable in acid (Thoumsin & Coussement, 1964).

The milled copper oxide material was submitted to a two-stage flotation process. The first step was known as the sulphide pre-flotation step, where the fraction of sulphide minerals present in the ore was separated from the predominantly copper oxide containing material. The copper sulphides concentrate from this flotation step reported to the copper sulphide circuit. Subsequently the pre-flotation tails were subjected to an oxide flotation step that produced a copper oxide concentrate that reported to the leach plant. As opposed to copper sulphides, copper oxides do not float naturally. Traditionally, copper oxides are floated by controlled potential sulphidisation (CPS) to sulphidise the surface of the copper oxide mineral. The sulphidisation is achieved by

addition of a sulphidisation agent such as, sodium hydrogen sulphide (NaSH). The addition of the NaSH reduces the oxidation reduction potential to the desired target value of between -450 and -550µV. CPS works well in a laboratory environment, in practice however the recovery of copper oxides is highly dependable on factors such as conditioning time, procedure of mixing and other variables that are present in the plant environment. Overdosing of a sulphidising agent such as NaSH leads to the suppression of copper oxide minerals, whilst the remaining sulphide minerals in the ore are heavily suppressed in the flotation step. As a results copper recovery from mine to cathode was not satisfactory as a large quantity of copper was lost in the tailings of the copper oxide flotation step.

The problem faced with the flotation of oxidised copper ores is that even with the addition of flotation reagents, the recovery of the oxidised copper minerals are low and have been observed to be highly variable. KOV has predominantly oxidised copper ore and mining the mineral resources from the open pit is considerably more economical than running the underground operation which produces predominantly copper sulphides. Consequently, the decision was made to build an FTAL plant which is capable of leaching flotation tails. The FTAL plant effectively allows for the copper oxide flotation process to be removed completely.

The following must be kept in mind if the process is to eliminate the oxide flotation step:

1. The advantage of an oxide flotation circuit is, that it reduces the mass and thus the volume of metal containing slurry that needs to be processed by the hydro refinery. This means that a smaller leaching and Counter Current Decantation (CCD) circuit with smaller process equipment is required, thus the capital expenditure to build such a plant is less.
2. During the process of flotation only a fraction of the gangue minerals is floated with the copper and cobalt containing minerals, resulting in a lower gangue acid consumption value.

The case for the FTAL plant then lies with the mineralogical recovery gained by modifying the copper flotation steps. This means that the capital expenditure to purchase larger process equipment as well as the additional processing costs of treating the un-

concentrated material needs to outweigh the losses due to the low oxide flotation recovery. To provide the reader with a simple justification for the FTAL plant the following calculation is used as the basis for the thesis. The targeted overall oxide flotation recovery at the concentrator was 80%, this value will be used as a best-case scenario. This means that after the concentration step there will already be a 20% loss of copper. The following parameters were used:

1. Feed rate to the concentrator 1450t/h.
2. Feed grade 3.45%.
3. LME A grade copper price at the time of calculation US$5 900-00 per ton.

Using these values, copper to the value of US$517 098 420.00 per annum was lost to the tailings disposal facility. The total budgeted expenditure of building a facility that was capable of treating 1450t/h of un-concentrated ore was approximately US$420 000 000-00, resulting in a payback period of less than a year, payback periods of 5-7 years are considered to be good.

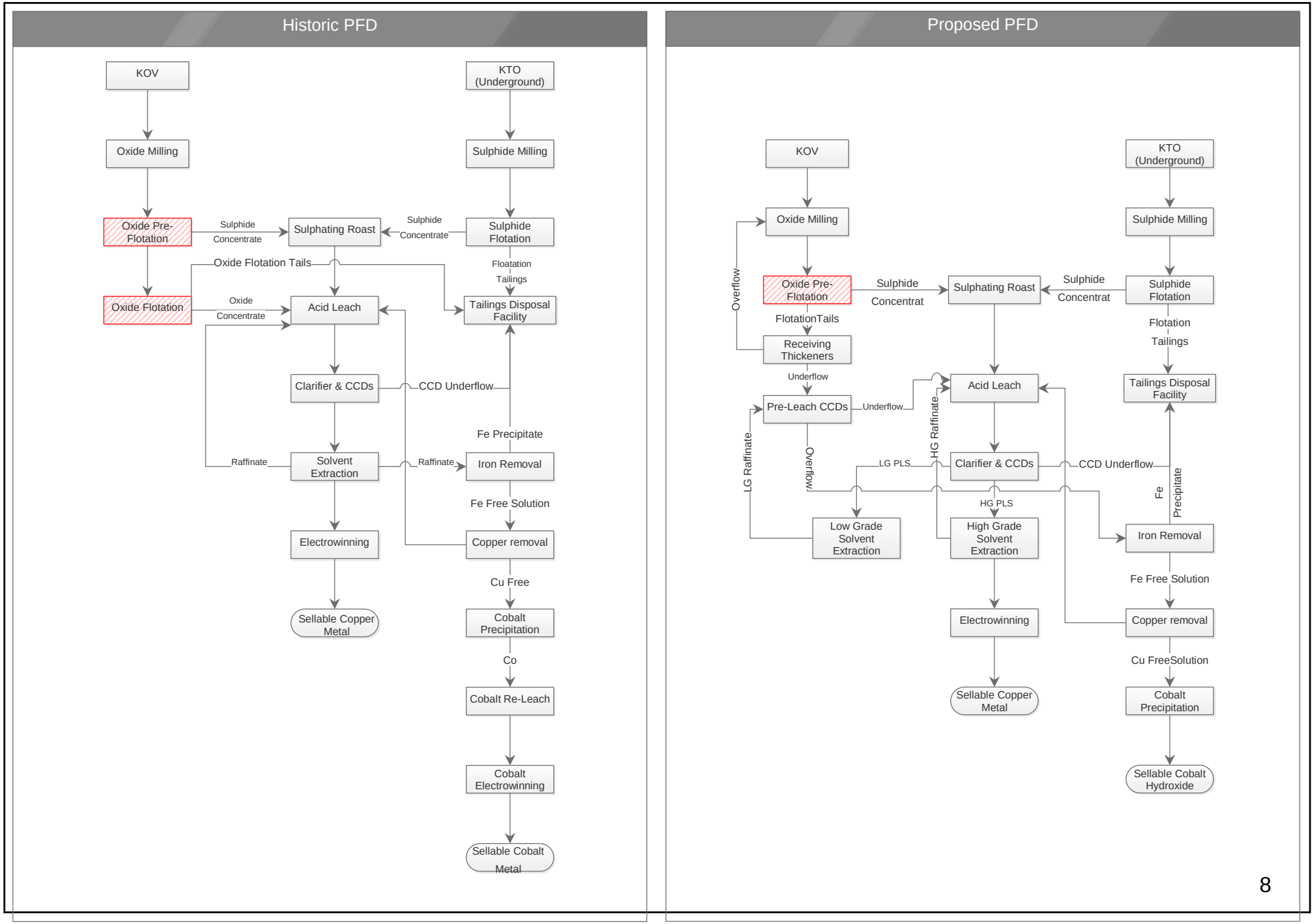


Figure 4: Historic (a) and proposed (b) processing circuits

1.5 Proposed operation circuit

The proposed FTAL plant, Figure 4 (b) is an agitated atmospheric acid leach, solvent extraction (SX) and electrowinning (EW) copper and cobalt hydrometallurgical processing facility (SENET, 2016). The future hydro refinery has been designed to process 9.3 MTpa of flotation tails from the concentrator as well as 200ktpa of sulphide concentrate which will be treated by the roasters that produce calcine from the copper sulphides. The hydro refinery is designed to produce 300ktpa of copper cathode as well as 22ktpa of cobalt hydroxide. The refinery is designed to have two parallel leach/CCD trains which carry 50% of the throughput each. The focus of this study was placed on the copper leaching circuit. Cobalt hydroxide was chosen as the sellable cobalt product mainly due to the battery industry preferring to purchase cobalt hydroxide product above cobalt metal. The SX and EW sections will only be mentioned briefly as required throughout the document.

The copper circuit consists of the following plant areas:

- Milling and flotation
- Displacement wash
- Leaching
- Post-leach thickening
- Counter-current decantation (CCDs)
- Neutralisation and neutralised tailings transportation
- Solvent extraction
- Electrowinning

1.5.1 Milling and flotation

The process from mine to metal starts at the open-pit and underground mines which feeds the mills dedicated to the copper oxide material mills and copper sulphide material mills respectively.

The milled copper sulphide material will be submitted to a conventional flotation process. A copper concentrate of between 34% and 38% will be produced. The concentrate then reports to a roaster where the partial sulphating roast creates a

copper product known as calcine that is readily soluble in acid (Thoumsin & Coussement, 1964).

The milled copper oxide material will be submitted to a single-stage flotation process, known as pre-flotation, where the fraction of sulphide minerals present in the ore will be separated from the predominantly copper oxide containing material. The copper sulphides concentrate from this flotation step reports to the copper sulphide circuit. The flotation tails will then report directly to the receiving thickeners in the displacement wash circuit at the hydro refinery.

From this point forward, feed refers to the flotation tails received from the oxide flotation step, unless when specifically referring to the copper sulphide concentrate.

1.5.2 Displacement wash

1.5.2.1 Receiving

The purpose of the receiving thickeners is to remove as much as possible process water from the flotation tails that is received from the concentrator. The feed from the concentrator is pumped into receiving tanks that distribute the feed to the receiving thickeners. The underflow from the thickeners, 55-60%$_{wt}$ solids, which is transferred into the pre-leach thickener mixing tank. The overflow from the receiving thickeners is returned to the concentrator.

1.5.2.2 Pre-leach CCDs

The purpose of the pre-leach CCDs is to displace water with low grade raffinate from the low grade SX circuit to reduce the acid consumption by using residual acid in the low grade raffinate to consume some gangue minerals and leach a small amount of copper. This also reduces lime consumption in the iron/aluminium/manganese (FAM) precipitation circuit. In this circuit, slurry flows counter current to the low grade raffinate solution used for washing of the slurry. The slurry from the washing circuit is then transferred to the leach area. The overflow from the washing circuit reports to the iron precipitation tanks.

1.5.3 Leaching

The purpose of the leaching step is to contact the copper minerals with a lixiviant, in this case sulphuric acid, to dissolve the minerals of interest and transport it into the liquid phase as dissolved metal salts. The leach circuit consists of two leach trains, each with 6 leach tanks. The partially leached slurry from the displacement wash circuit is pumped to the leach storage tanks where it is brought into contact with the acidified HG raffinate and calcine from the roasters. The leach slurry is then pumped to the leach tanks where the pH is to be controlled using acidified raffinate. The leach slurry is then pumped to the post leach thickener.

1.5.4 Post leach thickening

The purpose of the post leach thickener is to separate the solid and liquid phases of the leached slurry stream. The target metals are now present in the liquid phase as dissolved metal sulphides along with other dissolved metals such as aluminium, magnesium, calcium and iron. The post leach thickener is the point of separation between the high grade pregnant leach solution (PLS) and the low grade PLS circuits that feed the high and low-grade solvent extraction circuits respectively. The post-leach thickener is fed from the leach tanks, the slurry is diluted using a forced dilution system to reduce the feed-well slurry density to 5-10%$_{wt}$ solids. Flocculent is added in the thickener feed-well as well as in the thickener feed pipe. The thickener underflow is pumped to the CCD circuit. The thickener overflow, pregnant leach solution (PLS), is pumped to the HG clarifiers, the clarified solution feeds the high grade SX trains.

1.5.5 Counter Current Decantation

The purpose of the counter current decantation circuit is to displace the base metal containing solution with acidified cobalt barren solution, to recover up to 99.5% of the PLS through the 7 CCD stages. CCD 1 is fed with the underflow slurry from the post leach thickener and the overflow solution from CCD 2. The importance of CCD 1 is that the metal containing solution from it, is the feed to the low-grade solvent extraction circuit. CCD 2 to 6 is fed with underflow from CCD (X-1) and overflow from CCD (X+1). CCD 7 receive slurry from CCD 6 and cobalt barren solution from the cobalt precipitation plant that serves as a washate solution. The underflow from CCD 7 is known as acidic tailings and is pumped to the tailings neutralization plant.

1.5.6 Solvent Extraction

1.5.6.1 High grade solvent extraction

The high-grade solvent extraction (SX) circuit is fed with PLS originating from the post-leach thickener. The purpose of the high-grade SX circuit is to selectively extract copper from the PLS, leaving behind any other dissolved metals. Extraction of the copper from the PLS is achieved by using an extractant that is synthesised to selectively bind to the dissolved copper ions. The copper containing organic extractant is called loaded organic. The high-grade loaded organic is then stripped in a subsequent step by contacting the loaded organic with an acidic solution called lean electrolyte. The barren organic is then recycled to the extraction stage. After the PLS have been stripped of the copper in solution, the acidic solution is now known as high grade raffinate, that is returned to the leaching area to be further acidified and used as a lixiviant.

1.5.6.2 Low grade solvent extraction

The low-grade solvent extraction circuit is fed with PLS originating from CCD 1. The purpose of the low-grade solvent extraction circuit is to selectively extract copper from the low grade PLS, leaving behind any other dissolved metals. Extraction of the copper from the PLS is achieved by using an extractant that is synthesised to selectively bind to the dissolved copper ions. The copper containing organic extractant is called loaded organic. The low-grade loaded organic is then stripped in a subsequent step by contacting the loaded organic with an acidic solution called lean electrolyte. The barren organic is then recycled to the extraction stage. After the low grade PLS have been stripped of the copper in solution, the acidic solution is now known as low grade raffinate, that is fed to the displacement wash CCDs.

1.6 Aims and objectives

The main aim of the tests conducted in this study is to determine if it will be profitable to leach flotation tails as received from the oxide pre-flotation stage from the concentrator. From the results obtained, a blending strategy will be developed and proposed for optimal copper leach efficiency and acid consumption.

Each of the parameters that has been chosen to evaluate has a theoretical impact on the copper leach efficiency as well as the gangue acid consumption.

By reducing the particle size, you effective create a larger reaction surface area for the lixiviant to make contact with the minerals, by doing this the chances of collisions between the solid and the liquid is increased. This could increase the rate of the leaching reaction. Smaller particles also usually mean that the minerals are more liberated, this could lead to better leach efficiency of copper and cobalt. As the leach efficiency for copper and cobalt containing minerals could increase as the particle size is reduced so is the extent as well as the rate at which gangue minerals, such as dolomite, are leached.

Increasing the pH is effectively increasing the concentration of one of the reactants in an elementary reaction. Increasing the concentration of one or more reactants will often increase the rate of reaction. This occurs because a higher concentration of a reactant will lead to more collisions of that reactant within a specific period of time.

An increase in temperature will raise the average kinetic energy of the reactant molecules. Thus, a greater percentage of molecules will have the minimum energy necessary for an effective collision to occur.

At a constant pH, a lower slurry density is expected to deliver a better extent of reaction. This is because there is a large quantity of reactant “A” that can collide with reactant “B”. Specifically, in the case of copper leaching, the liquid phase can become so saturated with copper ions in solution at high percentage solids that the reaction could become diffusion layer limiting.

To determine the effect of these variables the following objectives have been defined:

1. Determine the effect of particle size on acid consumption and the copper recovery of oxidised copper minerals.
2. Determine what the effect of pH, % solids and temperature have on acid consumption and the recovery of oxidised copper minerals.
3. Determine the effect of cobalt and gangue minerals on the leaching efficiency and kinetics of copper oxides.
4. Determine a blending strategy based on the current resource model.

It was an important part of this study to do all tests using samples that were representative. This meant that using a blend of core samples that would represent the ore that will be fed into the flotation and subsequently to the acid leach plant.

Some of the obstacles with the open pit was that it was large and consisted of multiple ore bodies namely, Oliveira, Variant, FNSR and Virgule. The ore bodies have been known to consist of varying ratios of the know lithologies in the orebody. This means that each ore body could have distinctly different leaching efficiencies due to the average copper and cobalt grade contained in the ore. Varying mineralogy across the ore bodies would also affect the leach efficiency as well as gangue acid consumption values. Each of the main ore bodies had a pre-estimated size, thus a blend could be determined that would accurately represent the open pit operation.

One of the advantages of mine was that it is a relatively old operation and the geologist were able to estimate with reasonable accuracy what grade could be expected from the mine in the short and long term by using mine development software and historical values. The exact ratio used in this study is discussed in more detail in the Materials and material preparation section. But the 4 core samples that would be used to create the blend was prepared in 1 meter sections, each meter was prepared as described in the Procedures section, and analysed as described in the Further Analysis section of this document. Using the analytical results for the core samples a theoretical blend sample chemical composition was calculated before the samples were blended. However, the calculations indicated that the blend would be representative, and no alterations was made to the blend ratio as received from the mine planners.

1.7 Research approach

The first step in evaluating the acid consumption values and the copper leach efficiencies, was to test core samples available at that time to create base line values. Core samples were received from FNSR, Oliveira, Virgule and Variant ore bodies in the KOV pit, all of which included the following lithologies; Ratlillas, Ratgris, DSTRAT, SDS, SDB, RSF and RSC.

The following steps were followed during the initial screening. The samples were crushed and milled to a particle size of P_{80} 150µm followed by a 4-minute flotation step to remove the copper sulphide minerals present in the sample. The concentrate from this step was submitted to the laboratory for analysis. The flotation tailings were dried and sent for chemical analysis and labelled as the leach feed.

The flotation tails were then re-pulped and vigorously agitated and leached at 30°C using diluted sulphuric acid, the pH was maintained at 1.5 for a period of 4 hours. After the leach process, the acidic slurry was filtered, and washed using 500mL of demineralised water that has been acidified to a pH of 2.5.

To determine the optimal leaching conditions, subsequent tests were performed by varying the particle size, pH, temperature and the % solids in the slurry and is described in more detail in the Experimental section under the Experimental plan heading.

The optimised leaching conditions were used to leach a large quantity of core samples to verify the results from the initial core samples. The leach conditions found to be optimal was also used to determine leach kinetics of copper oxide minerals. One of the aims is to develop a model to assist in creating an ore blending strategy for the daily blend from the ore stockpiles.

It should be noted that the proposed flowsheet in Figure 4 includes a sulphide flotation circuit that feeds a pair of sulphating roasters. The calcine product formed by the roasting process could contributes to the acid balance. The leaching the copper from

copper sulphide concentrate is at the very least net acid consumption neutral. It is likely that the calcine will decrease the amount of acid that needs to be trucked to the leach plant. Evaluating the effect of calcine was not possible because at the time of the study the roasters were not in operation anymore and obtaining calcine samples would not be possible. Secondly, the refinery plant planned to start its roasters only two years after the FTAL plant was completed, the concentrated sulphide from the pre-flotation step would be stockpiled and reclaimed once the underground operation was re-started.

The effect of the minerals in the ore was analysed by identifying 3 distinctly different core samples based on their leaching results. When there is a presence of multiple metal ions in a system the reaction mechanisms can become complex due their electron charge. The basic and well-known process of copper cementation with iron filings could apply if the right combination of minerals is present in the leach system. This could lead to a decrease in copper leach efficiency as some of the already leached copped ions in solution could precipitate out as copper metal. Mineralogy will also reveal if there are any correlation between gangue acid consumption and well-known carbonate minerals which are known to regularly consume acid.

2 Literature review

2.1 Flotation

Many copper deposits contain significant amounts of oxidised copper minerals, such as malachite, and chrysocolla. These minerals are rich in copper, with copper weight percentages of 57.5% and 33.9% respectively. Copper oxides are regularly not reclaimed because they are not easily recovered by conventional thiol collection techniques effective for sulphide minerals.

2.1.1 Principles of operation

Froth flotation, patented in 1906, was originally employed to concentrate copper sulphide minerals and later refined to treat a host of other minerals such as malachite, hematite and other oxides copper minerals. Flotation uses physical chemistry to concentrate valuable minerals to a degree that makes the development of a mine and extracting the minerals from the ground economically justifiable. The main mechanism of flotation is the selective attachment of a mineral to an air bubble, caused by the minerals' hydrophobic nature. Other factors of the flotation process include mineral entrainment and/or entrapment. Due to these two phenomena a single stage flotation step is usually not sufficient for upgrading the mineral concentration, and flotation circuits are employed (Wills & Napier-Munn, 2006).

The process of froth flotation relies on the surface properties of the mineral. Copper sulphides are naturally hydrophobic in an oxidative environment and attach to air bubbles easily, however, the addition of reagents is required to temporally alter the surface characteristics of copper oxides to adhere to the bubbles (Lee, et al., 2008). A typical copper flotation cell can be seen in Figure 5. The process of flotation only works with fine to very fine particles as the force due to the weight of the particle should not be greater than the force of adhesion between the bubble and the particle caused by the difference in their polarity (Wills & Napier-Munn, 2006).

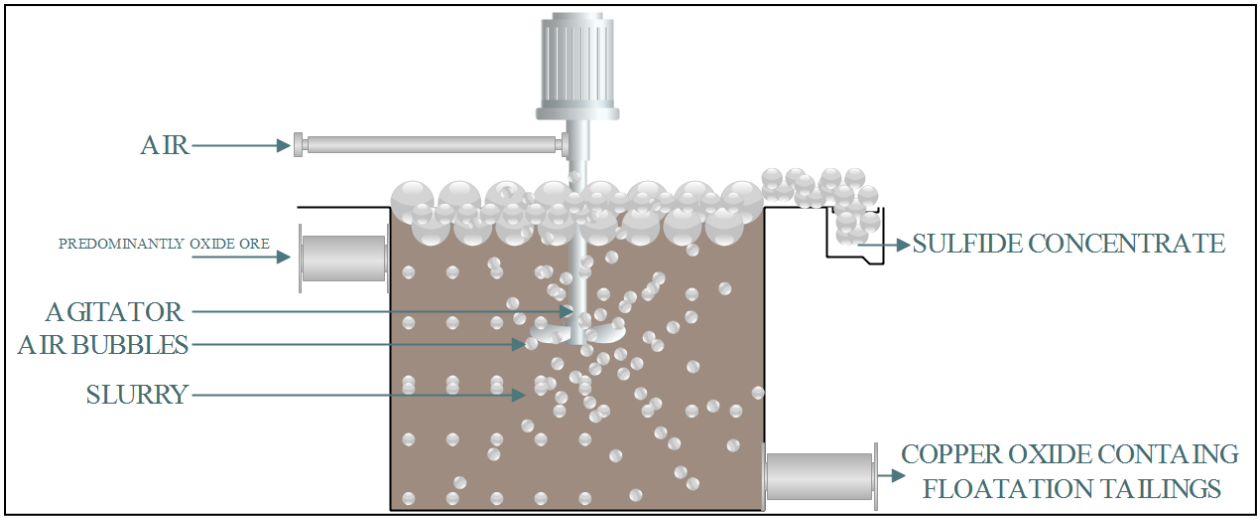


Figure 5: Froth flotation of predominantly copper oxide containing ore

Once a mineral has attached to an air bubble and the bubble has reached the surface of the flotation cell the bubble must be kept intact, i.e. the bubble must not burst, otherwise the minerals will simply fall off and sink slowly back to the bottom of the flotation cell. To maintain the physical integrity of the bubbles a reagent is used, generally known as flotation reagents (Wills & Napier-Munn, 2006).

2.1.2 Reagent addition

Flotation reagent can be broken down into two groups namely collectors and frothers. The purpose of collectors is to leave the surface of the mineral water repellent or hydrophobic to an extent that the mineral will adhere to the air bubble upon contact. Collectors are added to the slurry in what is known as a conditioning tank, that allows the collector to adsorb to the surface of the mineral. The collector's function is to reduce the stability of the hydrated layer surrounding the mineral (Wills & Napier-Munn, 2006).

In a typical oxide flotation process, a sulphidisation step would be required using a reagent such as sodium hydrogen sulphide (NaSH) to regulate the slurry Eh within the values of -450 and -550 mV. The risk with sulphidisation is that the addition of the reagents needs to be monitored and controlled very carefully. As under-dosing will result in an inefficient sulphidisation process, and over-dosing of NaSH would suppress the flotation of oxidised copper minerals such as malachite (Phetla & Muzenda, 2010).

The sulphidisation reaction is a heterogeneous reaction with two secondary reactions. The first step is the formation of a primary sulphidised layer on the surface om the oxide mineral. The adsorption of sulphur to form a primary sulphidised layer occurs rapidly. The sulphidised layer is formed as sulphide ions come into contact with copper oxide and react to form copper sulphide. The secondary sulphidisation processes includes the formation of a secondary copper sulphide layer which includes the formation of an oxysulphide species. The formation of the secondary sulphidised layer takes place as copper ions diffuse through cracks in the primary sulphidised layer.

Sulphidising agents are typically a type of sodium sulphide. Sodium sulphide enhances the flotation of oxidized copper minerals with xanthate collectors. Equation 1 is an example of the net sulphidisation reaction where M^{2+} is the surface metal ion and A^{2-} is the anion resulting from sulphidisation (Davidson, 2009).

$$M^{2+}A^{2-} + 2Na^{+} + S^{2-} \rightarrow M^{2+}S^{2-} + 2Na^{+} + A^{2+} \qquad \textit{Equation 1}$$

The most commonly employed collector for the flotation of copper oxide minerals, especially malachite, is of the xanthate family, such as the sodium n-butyl xanthate (SNBX) used as a collector at the concentrator under investigation, see Figure 6.

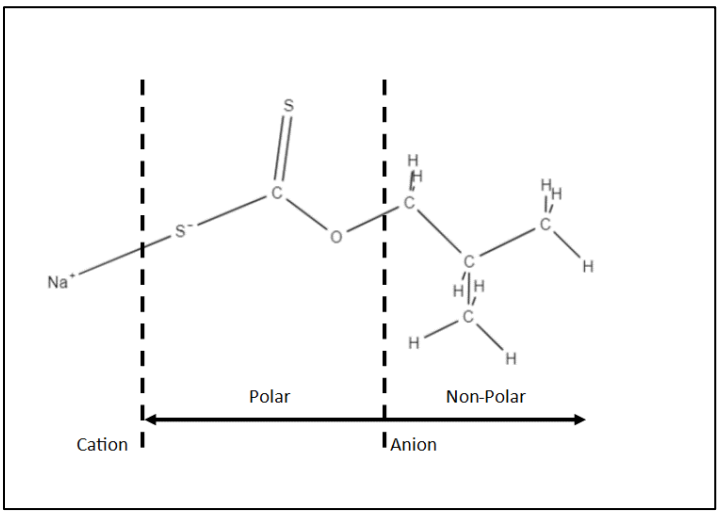


Figure 6: Molecular structure of sodium butyl xanthate

Once the slurry has been conditioned, a frother is used to aid in the stability of the bubble formation. A frother should have the ability to adsorb to the air-water interface and reduce the surface tension, to prevent the bubble from popping. Many frothers have similar structures as collectors and contribute to collection of minerals but could lead to the build-up of froth on thickeners, which is regularly experienced in concentrate-leach circuits, so ideally a frother should not have a long lasting effect and should be soluble in water (Wills & Napier-Munn, 2006). Thus, the preferred choice of collector is an alcohol based collector such as SASFROTH G41.

The pH of the flotation circuit also has been shown to play a vital role in the selectivity of the mineral being floated as well as the oxide flotation recovery. It has been found that recovery of copper sulphides can vary between 73% to 82% for pH values of 7 and 11 respectively (Gharaei, et al., 2014). Because the use of lime has no effect on the flotation of copper minerals it is employed to regulate the pH of the system due to its low cost as well as being regularly available.

pH also plays an important role in the oxide flotation process, because hydrogen ions compete with surface metallic sites to react with collector anions. Hydroxide anions compete with the collector for surface metallic sites. pH levels affect the dissolution of the copper oxides. Hydrogen ions force copper ions into solution by exchanging into the crystal lattice. Copper ions in solution, are chelated by the collector. This process depletes the collector and effectively stops the process of flotation. Typically, copper oxide flotation occurs at a pH between 8 and10.

2.1.3 Typical froth flotation circuit conditions

In a mixed oxide-sulphide mineral deposit as found in weathered ore bodies there is a small but economically justifiable amount of copper sulphides that can be recovered by froth flotation and separated from the oxides (Davidson, 2009). The main reason for the separation is driven by the fact that copper sulphides do not regularly dissolve in acid and need an additional treatment step before it can be leached, a step such as roasting for example. Certain circuits may even have separate refining circuits for the oxide and sulphide streams such as a smelter for sulphides and a leach/SX/EW circuit for their oxide minerals.

Typical grinding required for flotation ranges from P_{50} to P_{90} 75µm (Wills & Napier-Munn, 2006) depending on the mineralogy and laboratory findings. Ideally, the flotation slurry needs to be as dense as possible, as it reduces the equipment size required to obtain a required residence time in the flotation circuit and it reduces reagent consumption. As most reagent's efficiency is highly dependent on the concentration of the reagent in solution. Typical copper flotation circuits operate with slurry composed of 25-40%$_{wt}$ solids. Typically, an increase in solids concentration will results in better recovery and lower concentrate grades.

To control the pH of the flotation circuit and maintain a basic environment above a pH of 7.5 typically 1-5kg of burnt lime per (CaO) ton of ore is used, 0.002-0.03kg/MT of xanthate and 0.02-0.15kg/MT of frother is used (Wills & Napier-Munn, 2006).

2.1.4 Effect of key variables

The key variables in froth flotation include:

1. Flotation time
2. Pulp density
3. pH of slurry
4. Eh of slurry
5. Reagent concentration

Increased flotation time will result in higher recovery of minerals, but also leads to a lower grade (Lee, et al., 2008), a typical grade recovery curve can be seen in Figure 7. Grade recovery curves are obtained by collecting flotation concentrate samples at pre-determined time intervals. In the beginning of the flotation process you will have a very high assay, but the recovery would be very low. The longer you continue with the flotation process the more minerals you would recover, however gangue minerals are also recovered in the process, this means that the concentrate grade decreases over time, as illustrated in the figure below.

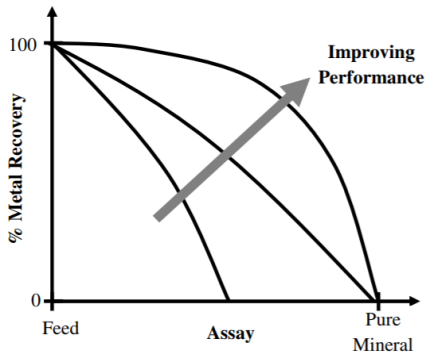


Figure 7: Typical flotation grade recovery curve redrawn and adopted from Kawarta, 2014

Gharaei et. al. (2014) found that pulp density showed that copper recovery had a linear directly proportional correlation to the pulp density and a second order inverse proportional effect on the grade of the concentrate, similar findings were made by Lee et al (2008).

The pH significantly alters the performance of the xanthate conditioning step responsible for sulphidisation of the minerals, preventing the hydrophobic layer forming around the mineral (Castro, et al., 1976). Tests by Castro et al. (1976) showed a significant decrease in xanthate uptake as a function of pH, as can be seen in Figure 8.

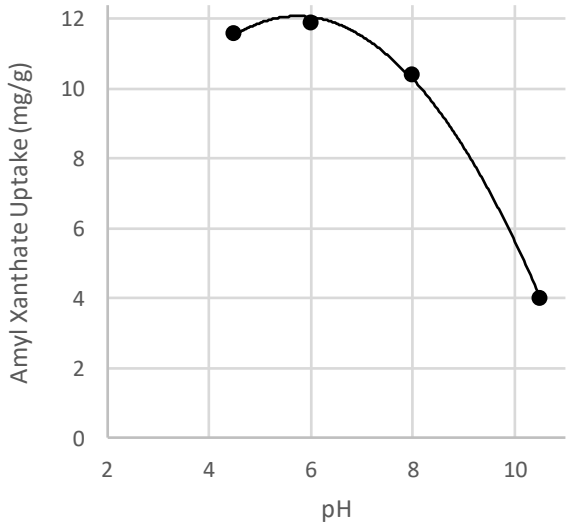


Figure 8: Influence of pH on xanthate consumption by chrysocolla redrawn and adopted from Castro, et al., 1976

Castro et al. (1976) also found that overdosing xanthates by having reagent concentrations by as little as 1g/L caused an irreversible depression effect on froth

flotation of copper minerals. However, methods of controlling reagent addition have been developed. Dielectric analysis measurements are quick and accurate and could be employed for effective reagent addition control.

2.2 Leaching

The goal of a hydrometallurgical process is to effectively and economically extract valuable components from ore. In this case the aim is to economically produce LME A grade copper cathode from oxidised copper minerals as well as copper sulphide concentrates, whilst having a minimal impact on the environment. The advantage of refining metals by hydrometallurgical methods is that it is possible to achieve good recoveries from low grade material, from as little as 0.4%wt Cu in ore.

The leaching of base metals can make use of a variety of lixiviants, depending on the desirable metal to be extracted. Copper leaching entails the dissolving of copper from minerals using a lixiviant to transport the metal into the liquid phase. In most cases sulphuric acid is used as the lixiviant.

2.2.1 Copper Leaching chemistry

Copper minerals are generally directly leached in sulphuric acid, these minerals include copper carbonates, oxides, hydroxy-chlorides, hydroxy-silicates and sulphates. These minerals are generally referred to as “oxides” (Davenport, et al., 2002), which will also be the convention followed in this thesis. The copper oxides usually form part of a matrix of other minerals such as quarts and dolomite and need to be liberated by first crushing and milling the ore. It was found in previous studies that the leach efficiency of copper starts decreasing for particles bigger than 212µm, test results from leaching studies on dolomitic coper ores showed the copper leach efficiency dropped from 93% to 80% for particle sizes of P_{78} 75µm and P_{80} 212µm respectively (Ntengwe, 2010), thus only particles sizes of 212µm and smaller was considered for this study.

Copper recovery will continue to increase up to a maximum of 100% as the particle size decreases, but leaching carbonate rocks such as dolomite, calcite and anchorite will also increase and consume acid following the chemical reaction seen in Reaction

1. The temperature and lixiviant concentration also play integral roles in the leaching efficiency, these effects were tested and reported by Ntengwe (2010) and is discussed in more details in the Copper leaching kinetics in chapter 2.2.2.

$$CaCO_3 + H_2SO_4 + H_2O \rightarrow CaSO_4 \cdot 2H_2O + CO_2$$ *Reaction 1*

It has been observed that making use of sulfuric acid is economically unfeasible in certain cases. This is due to calcium-magnesium carbonate minerals that regularly dissolve in acids (Tanda, et al., 2016). The calcium carbonate reaction is shown in Reaction 1.

Carbonate gangue minerals such as calcite, dolomite and ankerite are often associated with oxidised copper ore deposits that lead to high acid demand during the leach process. Alarmingly carbonate gangue minerals virtually has an acid consumption to mineral weight ratio of 1:1. It was reported that calcite and dolomite consumed 930 and 980 kg of sulphuric acid per ton of the mineral (International Atomic Energy Agency, 2001).

Minerals containing copper and cobalt which are regularly found in KOV include malachite, chrysocolla, hetrogonite and pseudomalachite. Copper sulphides like chalcopyrite, bornite, chalcocite and carrollite do not easily leach or don't leach at all unless they are exposed to a sulphating roast at temperatures above 660°C (Davenport, et al., 2002). The roasting reactions for the most common copper sulphide minerals in the Katanga province namely chalcocite, carrollite and chalcopyrite are seen as Reaction 2, Reaction 3 and Reaction 4 respectively (Thoumsin & Coussement, 1964):

$$2Cu_2S + 5O_2 \rightarrow 2CuO \cdot CuSO_4$$ *Reaction 2*

$$4Co_2CuS_4 + 27O_2 \rightarrow 8CoSO_4 + 2CuO \cdot CuSO_4 + 6SO_2$$ *Reaction 3*

$$Cu_2S \cdot Fe_2S_3 + 7O_2 \rightarrow CuO \cdot CuSO_4 + Fe_2O_3 + 3SO_2$$ *Reaction 4*

There are numerous existing mechanisms that describe leaching, two of which are widely used.

The first mechanism uses the solubility of a solid in a liquid, while the second involves chemical reactions that involves the mineral and the lixiviant. The process of leaching minerals become increasingly complex due to the formation of by-products like gypsum that may form a physical barrier between the lixiviant and the base metal containing mineral, resulting in passivated leach kinetics. In the case of chrysocolla for instance, a hydrated layer of silica is believed to form as the copper is leached, restricting any further leaching (Tanda, et al., 2016).

The leaching reactions of oxidised copper minerals namely azurite, malachite, tenorite and chrysocolla is given by Reaction 5 to Reaction 8 respectively:

$$Cu_3(CO_3)_2(OH)_2(s) + 3\,H_2SO_4(aq) \rightarrow 3\,CuSO_4(aq) + 4\,H_2O + 2\,CO_2 \qquad \textit{Reaction 5}$$

$$Cu_2(CO_3)(OH)_2(s) + 2\,H_2SO_4(aq) \rightarrow 2\,CuSO_4(aq) + 3\,H_2O + CO_2 \qquad \textit{Reaction 6}$$

$$CuO(s) + H_2SO_4(aq) \rightarrow CuSO_4(aq) + H_2O \qquad \textit{Reaction 7}$$

$$CuSiO_3.2H_2O + H_2SO_4 \rightarrow CuSO_4 + SiO_2 + 3H_2O \qquad \textit{Reaction 8}$$

2.2.2 Copper leaching kinetics

The effect of temperature and reactantant concentration on the leach efficiency of dolomitic copper ores were investigated by Ntengwe (2010) and it was found that at a pH of 1.8 the leach efficiency increased by 20% with an increase of temperature from 25°C to 60°C. From Ntengwe's (2010) data there was only a significant increase of leach recovery from temperatures exceeding 50°C. In other studies where the leaching of malachite was investigated an increase in temperature from 25°C to 80°C increased the leach efficiency from 94% to 98% (Bingol & Canbazoglu, 2004).

The concentration of acid on the leach efficieny was also investigated and found that the leach efficiency varied from 79% to 91% for pH values 3.0 and 1.8 respectively. This could be expected as the number of possible colissions of reactants are proportional to the concentration of molecules in the system. The concentration dependance on the rate of reaction is given by Equation 2 for an elementary reaction

that involves two reactants namely A and B. With $-r_a$ being the rate of the reaction with regards to chemical A, C_a and C_b being the concentration in mol per liter of reagents A and B respectively. k is known as the rate constant,

$$-r_A = kC_aC_B \qquad \textit{Equation 2}$$

Whenever a heterogeneous reaction system is investigated there are two additional factors that should always be kept in consideration, in addition to the normal consideration of temperature and reagent concentration (Levenspiel, 1999):

1. The mass transfer between phases
2. Contacting patterns of the reacting phases, such as plug or mixed flow in co-, con- or cross- current reactors.

Various models have been developed to describe heterogenous chemical reaction environments, such as the agitated leach system which is studied in this thesis. The kinetics model chosen to interpret the results of this study is the shrinking core model (SCM), because the SCM interprets the actual behaviour of real particles more accurately than other existing models (Levenspiel, 1999).

The SCM is explained as following these 5 steps, and the shrinking core is illustrated in Figure 9:

1. Diffusion of a reactant or lixiviant, in this case sulphuric acid, through the diffusion layer.
2. Penetration of the lixiviant through the layer of reacted material to the unreacted core containing un-leached metal containing minerals.
3. The lixiviant reacts with the unreacted shrinking core at the surface of the unreacted core.
4. Diffusion of the product through the layer of reacted material to the surface of the particle.
5. Diffusion of the product through the diffusion layer into the main body of leachate.

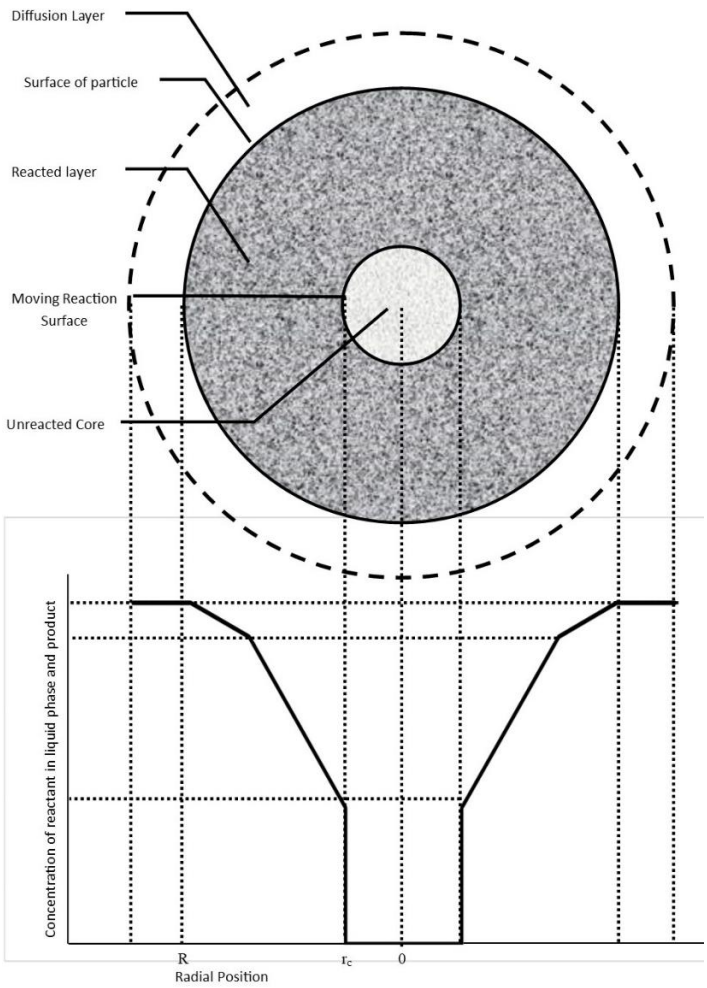


Figure 9: Representation of concentration of reactants in a typical shrinking core model

2.2.3 Modelling leaching systems

Weibull developed an empirical description widely adapted by the pharmaceutical companies around the world to describe the dissolution or leaching process of a chemical (Costa & Lobo, 2001). The model can be described by Equation 3:

$$m = 1 - exp\left[\frac{1-(t-T_i)^b}{a}\right] \qquad \textit{Equation 3}$$

Where *m* is the cumulative fraction of the solid in solution at time *t. b* can be found from the slope of the curve -ln(1-*m*) and T_i is the lag time before the reaction starts, this is normally 0. *a* can either be determined as the ordinate value of the reaction at *t* = 1, or *t* = T_b, with T_b, being the time of 100% reaction completion.

If the key variable for the change of the slope of the curve -ln(1-*m*) could be determined, this easy model could describe the leach efficiency achievable in the system.

This could typically be the influence from gangue minerals or other metals influencing the extent of reaction in the leach reactor. Thus making *b* a function of the key variable to achieve the desired model accuracy.

2.2.4 Typical atmospheric acid leach conditions

Approximately 20% of the world's copper is produced by hydrometallurgical processes (Davenport, et al., 2002). The hydrometallurgical process can be categorised into the following 3 steps:

1. The dissolution of copper and other acid soluble metals into a solution using sulphuric acid, this solution is known as pregnant leach solution (PLS).
2. Transferral of the copper from the PLS to a pure copper containing electrolyte, this is usually achieved by the solvent extraction (SX) process.
3. Electroplating, or electrowinning (EW), the copper in solution to produce pure copper metal sheets, known as copper cathodes.

This process is regularly abbreviated in literature as L/SX/EW. The leach process is found commonly as a heap leach process that relies on the same chemistry as agitated tank leach, but the equipment used to produce PLS is very different as the name suggests.

The particle size required for an agitated acid leach plant varies as the mineralogy changes, and different metals could be of importance, the ideal particle size is usually confirmed by metallurgical testing done on samples from the ore body. Table 1 gives a list of operations with their respective particle grind sizes (Dreisinger, 2009) (Dawson Metallurgical Laboratories, 2007):

Table 1: Different particle grind sizes used at copper plant around the world (Sole & Tinkler, 2016)

Operation	Particle size µm P_{80}	Operating pH
Boleo Copper-Cobalt-Zinc Manganese Project	38	
North Met Project	100-125	
Tenke Kwatebala Project	200	1.8

For a project focused on the extraction of copper, it can be found from the phase diagram for copper in the presence of iron that copper no longer exist in solution at pH values greater than 4 (Davenport, et al., 2002). Table 2 gives the average operating conditions as found across the world for copper leach plants:

Table 2: PLS compositions and pH values of PLS around the world (Sole & Tinkler, 2016)

Location	Cu in PLS (gpl)	pH
North America	0.3 - 3	1.5 - 2.5
South America	1.0 - 6.5	1.1 - 2.5
Africa	3.0 - 43	1.5 - 2
Rest of the world	1.0 – 40	1.0 – 2.5

Various studies indicated that temperature did not have a significant effect on the leaching of copper from ores originating from the copper belt (Ntengwe, 2010, Stuurman, et al., 2014). Which is a clear indication that the leaching of copper is not a reaction limiting liquid-solids system but rather depends on the product layer or diffusion layer limiting factors (Levenspiel, 1999). As a result leach plants in the area area operated at ambient conditions, but as a result of the exothermic nature of the leach reaction, temperatures of 40 to 60°C are not uncommon in practice.

2.2.5 Effect of key variables

2.2.5.1 Agitation Rate

The dissolution of malachite was investigated by Bingol et al.(2004), they found that only at 0 agitation could an effect be seen on the recovery of copper from the mineral. From as little as 150 rpm, the recovery remained largely unchanged at 87%, Figure 10 indicated their findings;

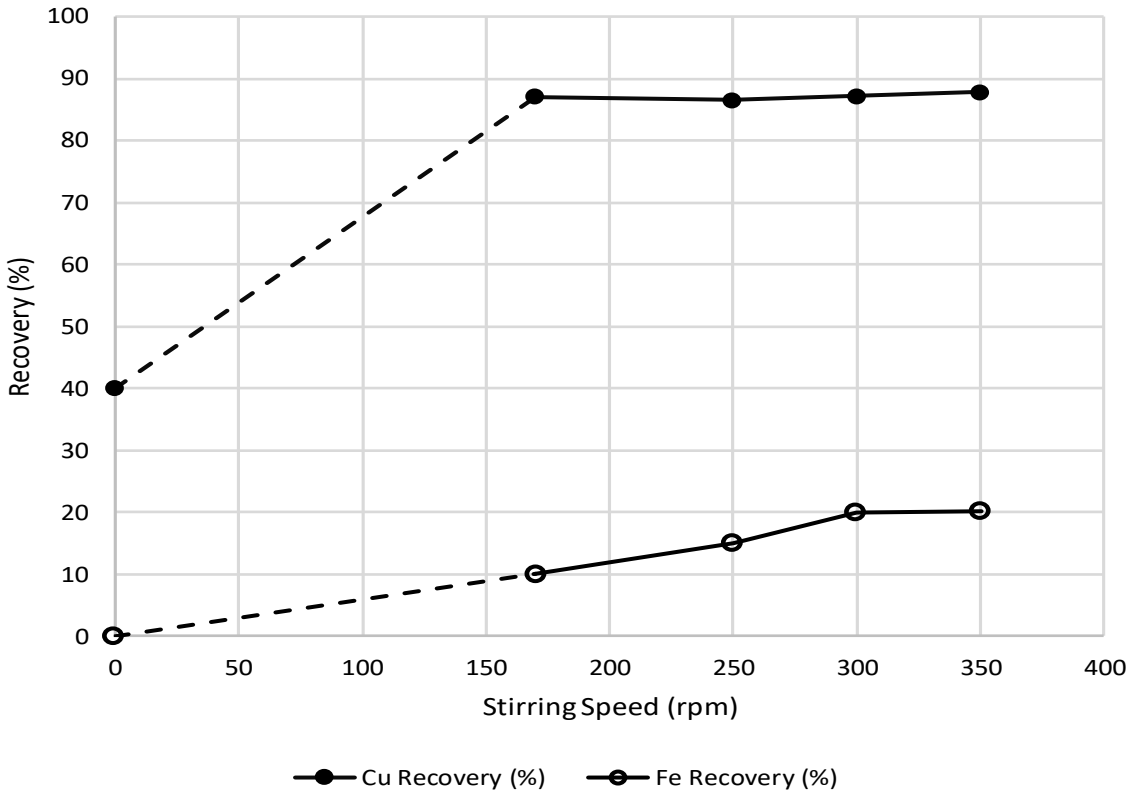


Figure 10: Effect of the stirring speed on the recovery of copper and iron from the malachite ore redrawn and adopted from Bingol & Canbazoglu, 2004

This would indicate that the reaction is not film diffusion limiting, but this is only applicable if the major copper oxide mineral is malachite. Similar findings were made by Shabani, et al. (2012).

2.2.5.2 Particle size

A reduction in particle size results in a greater surface area for the lixiviant to react with the mineral, this is true for copper, but also for gangue minerals such as dolomite or manganese and aluminium containing minerals that will be leached in parallel with copper, resulting in higher gangue acid consumption values that erodes the profits of the refinery.

Shabani et al. (2012) found that the optimum particle size for the leaching of malachite was between 105 and 150µm. In another study, particles greater than 212µm showed a significant decrease in copper leach recovery from malachite (Bingol & Canbazoglu, 2004), the results can be seen in Figure 11.

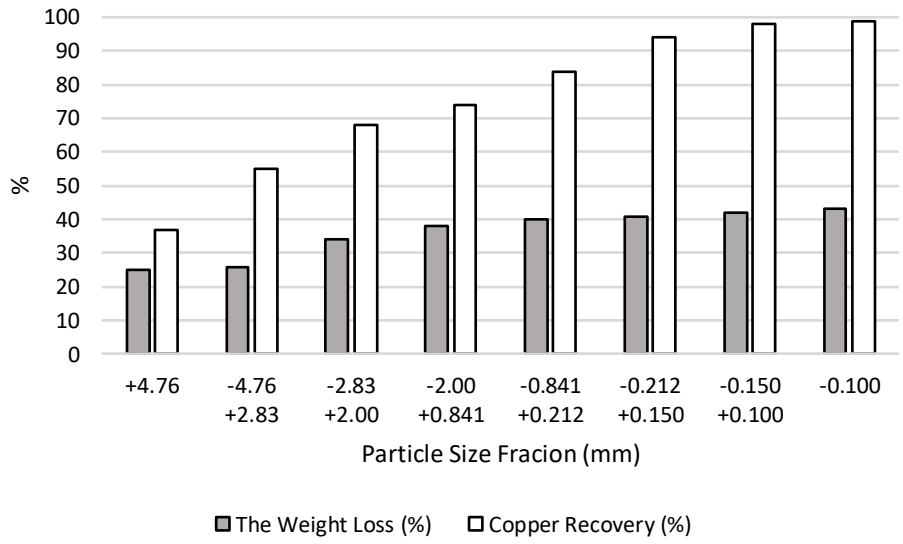


Figure 11: Effect of particle size of the crushed ore for the dissolution of copper from malachite redrawn and adopted from Bingol & Canbazoglu, 2004

2.2.5.3 Acid concentration (pH)

Stuurman et al. (2014) found that copper leach recovery dropped significantly only once pH values reached approximately 2. A pH of 2 corresponds to an area in the copper Eh-pH diagram where copper is no longer in solution but precipitates out as $CuFeO_2$. They also found that at very high acid concentrations, at pH values < 0, the leach efficiency once again decreased, possibly due to an acid impenetrable product layer forming on the particle surface.

Studies on malachite found that the concentration of citric acid on the leach efficiency of copper from malachite did not have a significant effect. Only when the test was conducted where the stoichiometric value of H^+ in the system was less than the copper available to leach did the leach efficiency drop (Shabani, et al., 2012), their findings can be seen in Figure 12;

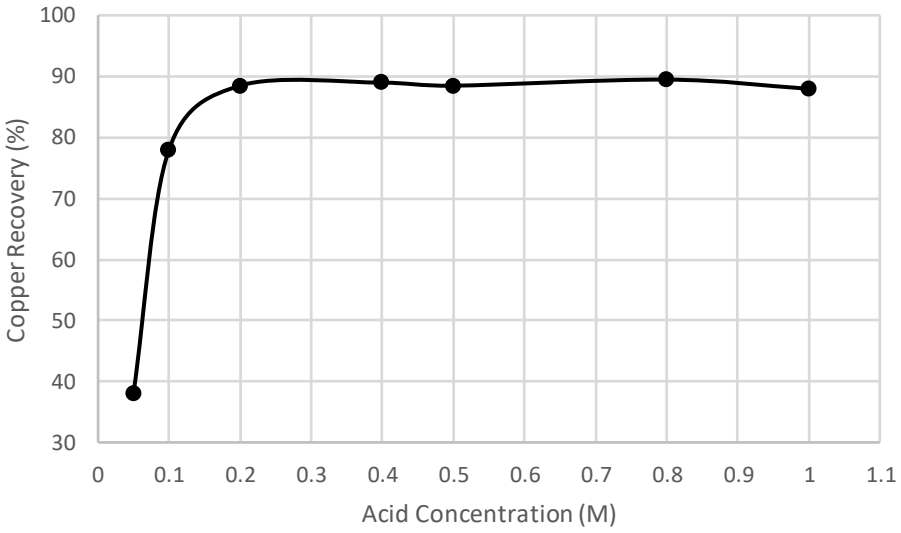


Figure 12: Effect of acid concentration on malachite leaching (S/L ratio=1:10 g/mL, ambient temperature ~20°C, time=1 h, stirring speed=400 r/min) redrawn and adopted from Shabani, et al., 2012

Bingol et al. (2004) also performed acid leach tests on malachite with sulfuric acid. These tests were also conducted using a method with a starting amount of acid and only showed low recoveries of copper once the amount of acid in the system dropped below the stoichiometric amount. Their findings can be seen in Figure 13.

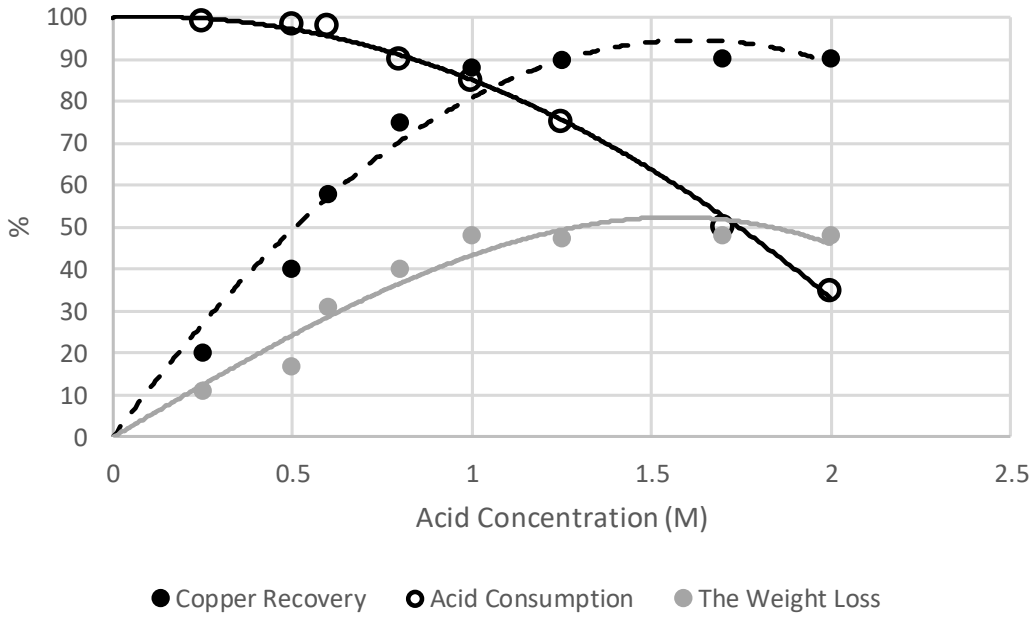


Figure 13: Effect of sulphuric acid concentration on the recovery of copper from the malachite ore, and on the acid consumption and the weight loss during the leaching redrawn and adopted from Bingol & Canbazoglu, 2004

2.2.5.4 Temperature

Stuurman et al. (2014) performed leach tests on ore from the DRC and found a marginal increase in copper leach recovery as the temperature was increased from 20°C to 40°C. Even though Arrhenius principle predicts increased reaction kinetics with an increase in temperature, it is also noted by Levenspiel (1990) that in a solid – liquid reaction system a small temperature effect on the kinetics indicates that the nature of the kinetics is not limited by the chemical reaction, but it's more likely to be product or diffusion layer limiting. The results obtained from Stuurman et al. (2014) can be seen in Figure 14.

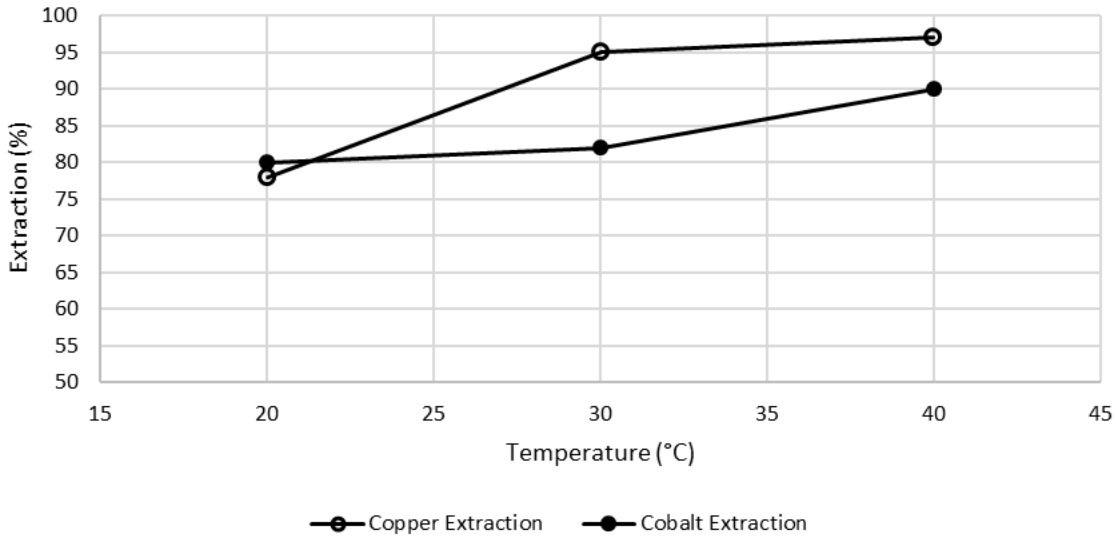


Figure 14: Effect of temperature on copper and cobalt dissolution redrawn and adopted from Stuurman, et al., 2014

Bingol et al. (2004) found similar behaviour of malachite in sulfuric acid, where the recovery was 94% and 98% for 25°C and 80°C respectively, their results can be seen in Figure 15.

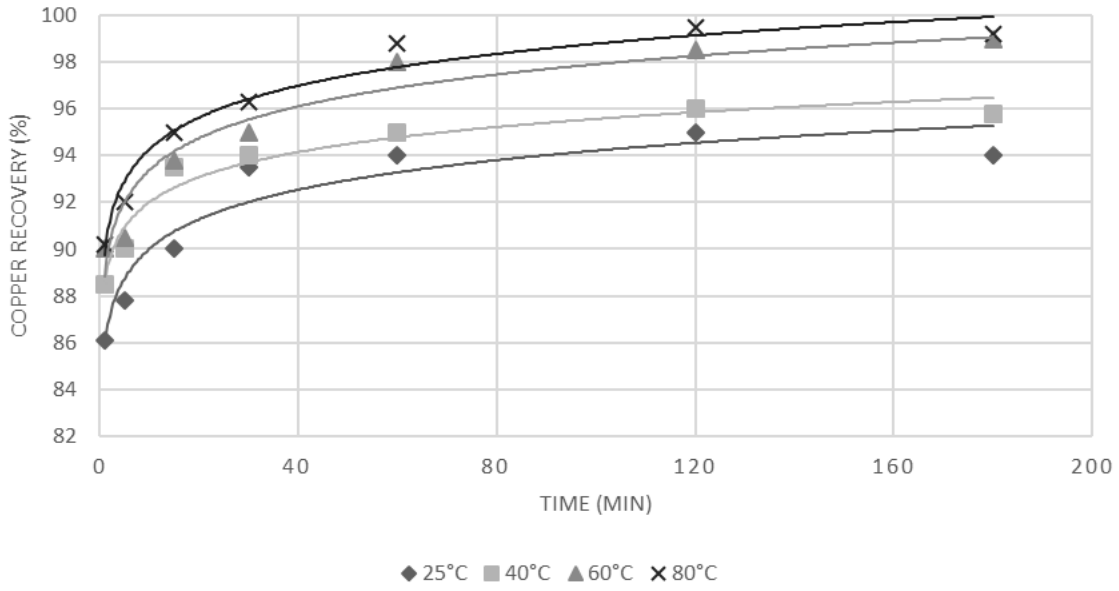


Figure 15: Effect of leaching time on the recovery of copper from malachite redrawn and adopted from Bingol & Canbazoglu, 2004

Similar findings were made by Ntengwe (2010), who studied the leaching of dolomitic copper oxide ore from the DRC/Zambia Copper belt, the results varied from 86% to 96% for 25°C and 60°C respectively, with a big drop in recovery to 66% at 15°C. The results can be seen in Figure 16.

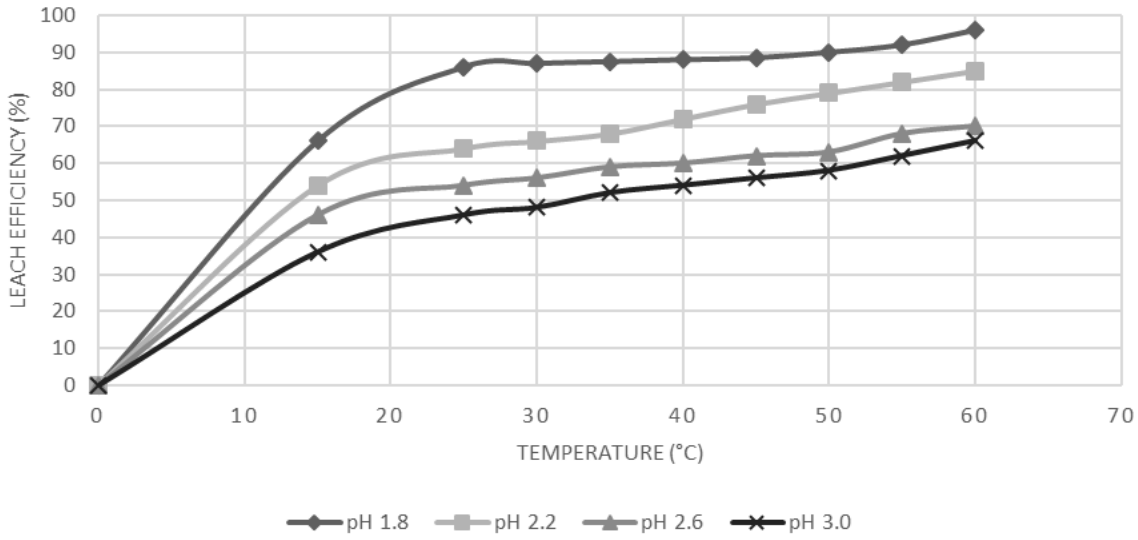


Figure 16: The effect of temperature on leaching efficiency of copper in sulfuric acid redrawn and adopted from Ntengwe, 2010

2.2.5.5 Solids concentration

Theoretically, higher concentrations of reactants should increase the amount of possible copper/H^+ interactions. This would lead to a higher concentration of copper in solution, i.e. a high-grade pregnant leach solution (PLS), not necessarily a better copper recovery from the solids. In addition, the acid might become the limiting reagent in the system and adding to the diffusion layer effect in the kinetics caused by a lower concentration difference on the particulate level. Thus, the ideal solids concentrations are normally found by metallurgical testing on samples of the ore body.

Acid leach tests conducted on malachite showed significant increases of leach recovery at solid to liquid ratios of 1:5 and smaller (approximately 15% solids or more). Once again due to the nature of the test, the extraction could be lower due to the acid in the system being the limiting reagent, their results are seen in Figure 17.

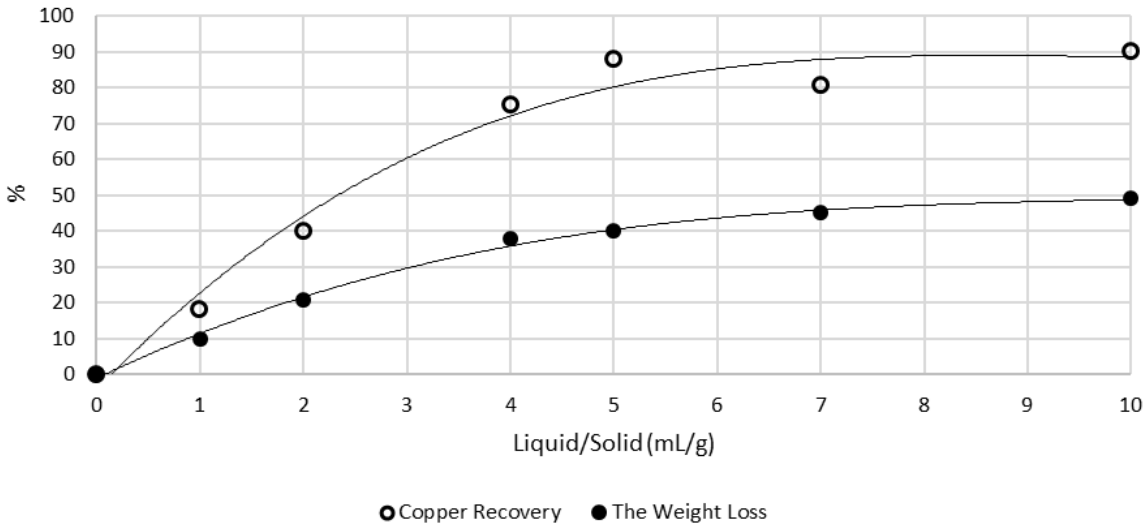


Figure 17: Effect of solid to liquid ratio (w/v) on the recovery of copper from malachite

3 Experimental

3.1 Experimental plan

From literature it has been found that the following factors affect the leach efficiency and gangue acid consumption figures:

1. Agitation Rate
2. Particle size
3. Acid concentration (pH)
4. Temperature
5. Solids concentration

Significant attention was placed on particle size and acid concentration, the reason being the financial implications involved when these two factors change. Milling operations are known to be some of the most cost intensive parts of any operation, but it is critical to maintain a balance between milling costs and metal losses caused by insufficient liberation. The mills at the operation was already existing but, the cyclones could be replaced if it could be proven that a bigger particle size would yield a profitable metal recovery. The cost of milling is closely followed by the cost of acid which is by far the largest operating expenditure of the hydro refinery. In addition to the cost, the reliability of acid delivery to the site was always questionable due to factors at the border between Zambia, where the acid came from, and the DRC. Thus the financial budget for acid and the motivation behind it always received a lot of attention from a corporate point of view.

From literature, previous experience at the refinery and knowledge from the sister company running a similar operation, the agitation rate has shown to have very little effect on the leach efficiency as long as all the particles in the slurry was suspended. No agitation or very little agitation will have adverse effects on the leach efficiency. All tests were conducted with the solid particles completely suspended and well agitated in 2L baffled stainless steel reactors by maintaining the stirrer speed constant throughout the tests at 350rpm.

To determine the effect of particle size on leach efficiency two geological field samples were used, one sample from the lithologies which represent the top layer of the open pit, namely *Ore Body Superior* (OBS) and another to represent the lithologies from the bottom layer of the pit, namely *Ore Body Inferior* (OBI). These samples were then dried, crushed and split. A 500g sample was used to create a milling curve. Samples to be leached were then milled to three grind sizes ranges of 75µm, 150µm and 212µm. Each size fraction will be leached in quintuplicate under the conditions seen in Table 3:

Table 3: Test conditions used to determine particle size effect

Temperature (°C)	pH	Agitation (rpm)	%Solids	Particle Size (P_{80})
30	1.5	350	30	75µm
30	1.5	350	30	150µm
30	1.5	350	30	212µm

All subsequent tests were done using a blend made from geological drill core samples. For these tests a blended composite sample was created based on a medium-term resource model from the mine engineers, the blend is explained in more detail in the Materials and material preparation section and in Table 6. To determine the effect of the weight percentage solids in the leach slurry a base test and two additional tests were done in duplicate under the leach conditions seen in Table 4:

Table 4: Test conditions used to determine percentage solids in leach slurry effect

Temperature (°C)	pH	Agitation (rpm)	%Solids	Particle Size (P_{80})
30	1.5	350	20	150µm
30	1.5	350	30 (Base)	150µm
30	1.5	350	40	150µm

From the particle size and %solids tests, the optimum leach conditions were identified and used to identify the effect that the acid concentration, measured as pH, had on the leach efficiency and acid consumption. For the pH effect tests, all samples were milled to a P_{80} of 150µm and samples were prepared at 30% solids, the pH effect was

tested by leaching the blended samples in duplicate under the conditions seen in Table 5:

Table 5: Test conditions used to determine pH effect

Temperature (°C)	pH	Agitation (rpm)	%Solids	Particle Size (P_{80})
30	1.0	350	30	150µm
30	1.5	350	30	150µm
30	2.0	350	30	150µm

All tests conducted to determine the effect of particle size, acid concentration (pH), temperature and solids concentration were done under a reductive environment to account for the effects of iron reduction as well as the reduction and leaching of cobalt in the ore. The reductive environment was created by addition of 200g/l sodium metabisulfite solution to control the Eh of the slurry to 375µS.

All experiments were done according to the procedure which will be discussed in the subsequent chapter, Procedures. Following the array of test mentioned above, more core samples became available, the reasons and details about the additional cores can be found in the Materials and material preparation chapter. From the new core samples 3 specimens were selected and sent for mineralogy. The specimens were selected based on specific criteria that is explained in the Materials and material preparation section, to explain the unique results obtained from the basic leach tests.

3.2 Materials and material preparation

At the conception of the project, a limited amount of drill core samples were available for analysis, however, with the assistance of the geological department, 4 core samples were identified as representative. The 4 core samples were from each of the ore bodies in KOV, namely; FNSR, Oliveira, Virgule and Variant.

Each core sample contained all the major lithologies namely; Ratlillas, Ratgris, DSTRAT, RSF, RSC, SDB and SDS.

Each lithology from each ore body was crushed and kept separate, thus there were a total of 28 individual samples.

The first phase of the study was to determine if the lithologies have significantly different characteristics, so they were leached at typical leach conditions taken from another similar project located 60km east of the mining and refining operation. The first phase was conducted at two grind sizes with a P_{80} of 150µm and 212µm. From other literature it was found that the exact definition of the grind size, or particle size used during the leaching tests and the method of how the sample of the particle distribution was obtained could be confusing, thus to avoid this in this thesis a clear explanation is justified.

All cores were first dried at 100°C for 12 hours, then crushed in a pendulum mounted jaw crusher with a closed side gap of 3mm. 500g of crushed sample was then screened on either a 150 or 212µm ro-tap to get an initial PSD, thereafter it was loaded into a ball mill filled with 285 steel balls of various sizes weighing ±20kg. The sample was then milled for 10, 50 and sometimes 120 minutes to obtain a milling curve. The samples were kept as a back-up and labelled as *"Milling Curve"*. A 500g crushed sample that was to be floated and leached was then milled for a duration of time as obtained from the milling curve. After milling the sample was screened with a 150µm or 212µm screen depending on which samples was being prepared, to confirm the grind of the sample. All samples within a P_{80} within a tolerance of 1% was accepted.

The second phase of the study was aimed at determining the effect that Temperature, pH and %solids had on the leach efficiency and acid consumption. For these tests a blended composite sample was created based on a medium-term resource model from the mine engineers, the following blend was used:

Table 6: Composite sample blending ratios

Ore Body	Lithology	Weight percentage
FNSR	RATGR	0.5%
FNSR	DSTRAT	1.6%
FNSR	RSF	2.9%
FNSR	RSC	8.0%
FNSR	SDB	6.1%
FNSR	SDS	0.7%
OLIVEIRA	RATGR	2.3%
OLIVEIRA	DSTRAT	3.6%
OLIVEIRA	RSF	5.3%
OLIVEIRA	RSC	19.5%
OLIVEIRA	SDB	11.1%
OLIVEIRA	SDS	2.5%
VIRGULE	RATGR	1.5%
VIRGULE	DSTRAT	3.7%
VIRGULE	RSF	5.5%
VIRGULE	RSC	14.0%
VIRGULE	SDB	10.1%
VIRGULE	SDS	1.2%

Phase 3 involved subjecting all the new cores that has been drilled to a standard "*leach procedure"* comprising of drying, crushing, milling, flotation and leaching each lithology from each core separately. This was done to create a profile of the ore bodies, to determine if there are any sections on the open pit that posed a risk in terms of low copper recovery and/or high acid consumption. Therefore, an additional 32 core samples were earmarked from the open pit mine for testing.

Three samples (Table 7) were selected for mineralogical analysis based on the following criteria:

1. The total contained copper had to be between 3 and 4%, based on the design refinery feed grade of ±3.5%.
2. They had to have distinctly different copper leach efficiencies based on the initial leach test.
3. At least one of the samples had to have a high gangue acid consumption.

The following 3 samples were selected:

Table 7: Table of samples selected for mineralogical analysis based on their initial laboratory leach results.

Fragment	Lithology	Head Grade TCu%	Head Grade TCo%	Leach CuOx Recovery	GAC
FNSR	SDS	3.31	3.09	88%	27
OLIVEIRA	SDB	3.53	0.65	95%	310
FNSR	RSC	4.07	O.19	99%	4

The purpose of these samples was to attempt to explain the abnormalities in their behaviour by analysing their mineralogy.

3.3 Equipment

3.3.1 Experimental setup

The combined experimental setup consisted of two parts, the first part is a Denver D12 benchtop flotation machine required to create flotation tailings that would be leached. The second experimental setup consisted of a benchtop leach setup that consisted of a water bath, overhead stirrers, pH measuring devices and an acid addition method. The detailed description of the setups can be seen in Equipment and Procedures sections.

3.3.1.1 Flotation

A D12, Denver laboratory bench scale flotation machine fitted with an electronic tachometer to measure the impellors speed in rpm was used to obtain the flotation tails that were subjected to the acid leach tests. A 2000ml flotation cell was used to float the milled and size verified samples. The flotation process is assisted using SNBX, 100g/t, and G41 frother, 30g/t. The Denver D12 benchtop flotation machine can be seen in Figure 18.

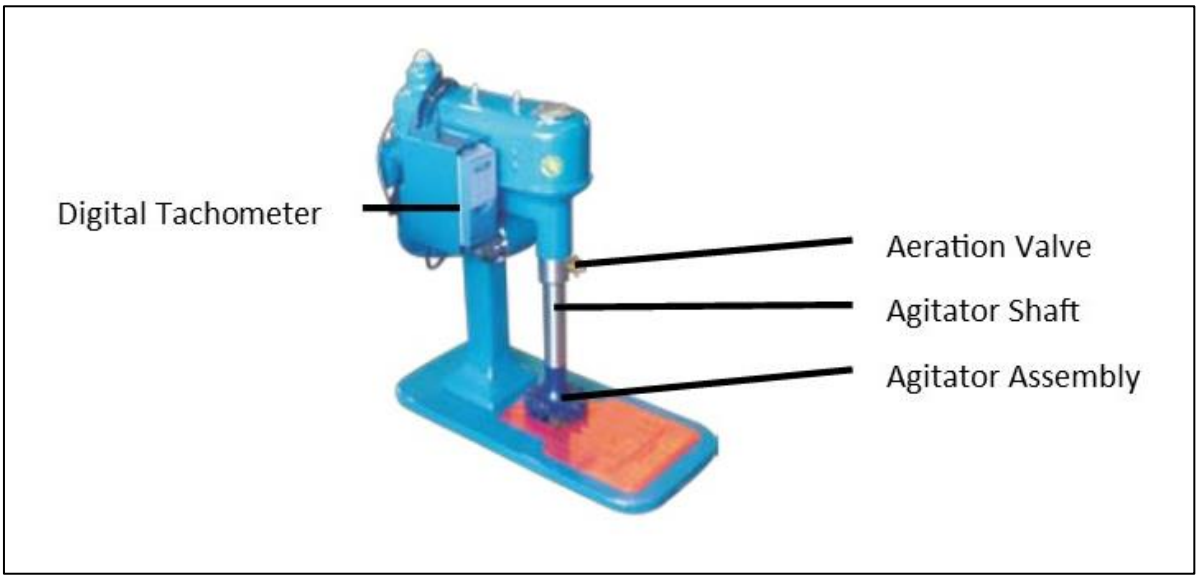


Figure 18: Setup of a Denver D12 flotation machine for laboratory flotation test work

3.3.1.2 Leaching

The leaching experiments were performed in 2000ml stainless steel reactors containing 4 baffles in each reactor to ensure adequate agitation. The slurry was agitated using Heidolph variable speed overhead stirrer fitted with digital tachometers. The overhead stirrers were equipped with 15mm high 3-bladed teflon agitators the blades on the agitators were pitched at 30°, as seen in Figure 19. The agitators were positioned 10 to 20mm from the bottom of the reactor during the leach tests. Control of the pH of the leach tests was accomplished by using 100ml ± 0.01ml burettes mounted above the reactors using laboratory stands and clamps.

The temperature was controlled using a specially manufactured hooded stainless-steel water bath fitted with an *FMH Instruments* temperature controller, water heating element and water circulation pump.

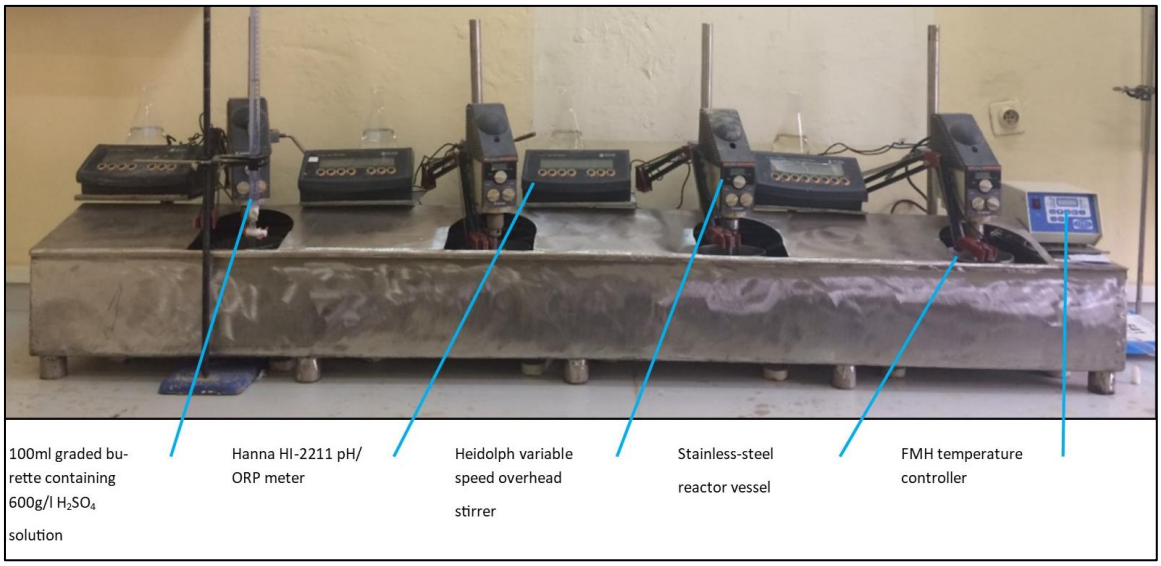


Figure 19: Experimental setup used for leach tests

The pH and temperature were monitored using Hanna HI-2211 pH meters each fitted with a HI-1131B pH electrode and a HI-7662 temperature probe. The pH probe and burette containing the acidic solution was placed on opposite sides of the reactor to avoid measuring the localised acid hotspots created adjacent to the acid addition point.

3.4 Procedures

3.4.1 Flotation Procedure

The flotation procedure followed for all samples was a standard procedure as developed previously at the concentrator. That was found to be optimal for the removal of relatively small amounts of copper sulphides found in the predominantly copper oxide mineral containing ore.

Once a sample has been milled and the P_{80} of the sample has been verified, the sample was subjected to a flotation step to simulate the removal of copper sulphide minerals.

All leach tests were done in duplicate, a 1000g (2 x 500g) solids sample was prepared to a slurry density of 1350g/L which is approximately 40%$_{wt}$ solids. The slurry would then be placed in the 2000mL flotation cell. The Denver D12 agitator assembly was lowered into the slurry, and the aeration valve was closed.

The agitator was started, and the impeller speed was set to 1200rpm by changing the settings on the digital speed controller located below the tachometer. The following reagents were then added at their respective dosages:

1. Collector (SNBX) at 100g/MT
2. Frother (G41) at 30g/MT

The slurry along with the collector was then allowed to condition for a period of 3 minutes. After the conditioning period the aeration valve was opened, and the concentrate was collected in 15s intervals using a froth paddle for a period of 4 minutes. To ensure the consistency of the froth collection, only a single stroke of the froth paddles were used every 15s, starting at the rear of the flotation cell and drawing past the agitator shaft and over the lip of the flotation cell. The paddles were then rinsed after each step with demineralised water using a wash bottle.

The concentrate sample collected were dried, weighed and sent for chemical analysis. The flotation tails were filtered, dried, weighed, split and sent for chemical analysis.

3.4.2 Leaching

Prior to leaching a large batch of diluted sulphuric acid was made up to a target concentration of 600g/L or 6.1 mol/L equivalent. Sulfuric acid was diluted to ±6.1mol/L in a 25L container that was fitted with a tap, see Figure 20. A large amount of heat was generated during the make-up of the solution. Therefore, addition of the concentrated sulphuric acid was achieved using a Watson Marlow QDOS30 pump with acid proof piping. The flow rate of the pump was set between 30 and 40 mL/min to ensure that the heat generated during the dilution of the acid would dissipate.

The pump was then stopped as soon as the desired weight of acid was added to the demineralised water.

Three 4ml samples of the diluted acid was further diluted by ratios of 2:1, 3:1 and 5:1 and sent to the analytical laboratory for a free acid analysis by using NaOH titration to determine the exact diluted sulphuric acid concentration. The average value of the three titration tests were used to determine the acid concentration. A wash solution for filtration was also prepared by adding 0.425ml of AR grade H_2SO_4 into 5L of de-ionized water to lower the pH to 2.5.

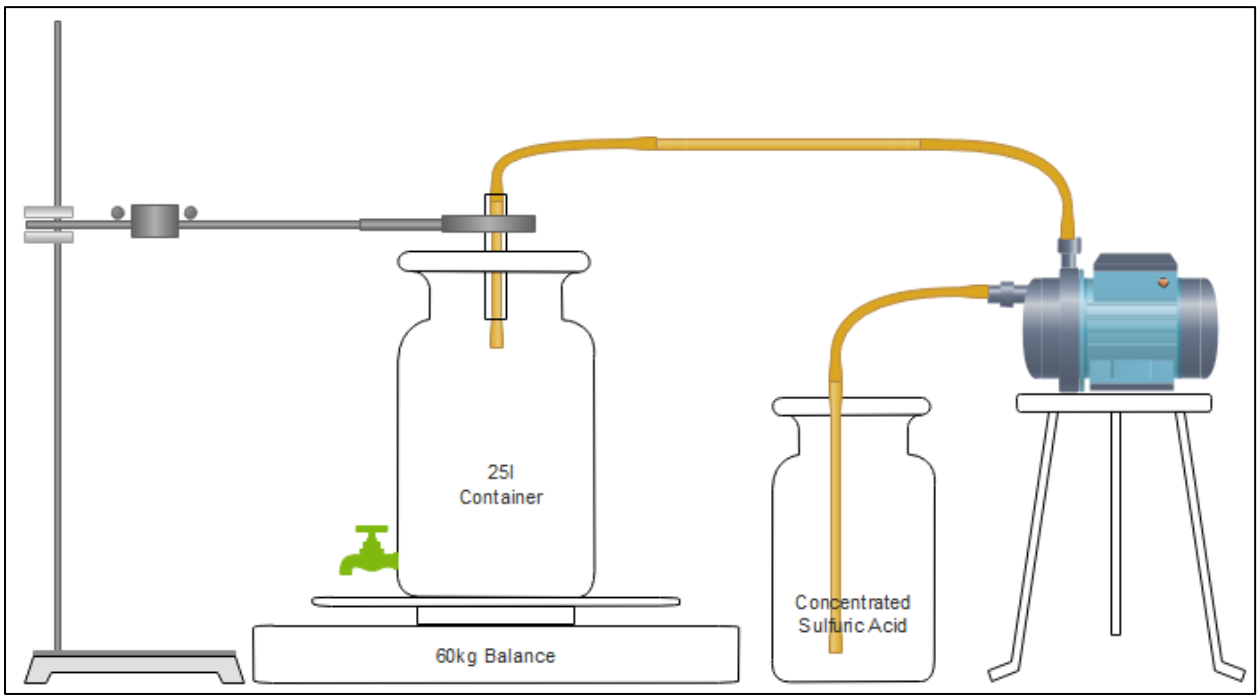


Figure 20: Experimental setup for diluting concentrated sulfuric acid

The dried tailings were then weighed and the required amount of water to create a 30%$_{wt}$ solids slurry was added into the stainless-steel reactors. The variable speed overhead stirrers were turned on and set to the desired speed of 350rpm. The water was allowed to reach the desired temperature in the temperature-controlled water bath. The water was acidified to the pH required for the test and the amount of diluted sulphuric acid used was noted.

Once the water has reached the desired temperature the solids sample was added to the solution and addition of acid immediately started. To control the pH at the desired value for a period of 4 hours, unless clearly stipulated that the leach test would be stopped after a different period of time.

The volume of acid added as well as the pH was noted at the following time intervals T(min) = 0, 10, 20, 30, 60, 90, 120, 150, 180, 210, 240. After 4 hours the samples were filtered, and the cake washed with the wash solution to ensure that there are no PLS remaining in the filter cake. The volume and weight of the filtrate and wash solution was measured and noted down. The wet filter cake weight was measured and noted. The filter cake was then dried, weighed, split and the solids samples were sent to the analytical laboratory for chemical analysis.

3.4.3 Further Analysis

The following analysis were done on the solid and/or liquid samples:

Mass pull analysis – Performed on all samples

Mass pull is defined as the weight of the concentrate produced divided by the total mass of the sample floated and can be seen in Equation 4.

$$Mass\ Pull = \frac{Dry\ weight\ of\ Concentrate}{Dry\ Weight\ of\ Sample} \qquad \textit{Equation 4}$$

Flotation recovery of contained sulphides – Performed on all samples

The flotation recovery of interest on the copper oxides treatment stream is the recovery of the copper sulphides present in ore. Thus, the recovery is a function of the copper sulphides (%CuS) in the concentrate divided by the total amount of copper sulphides in the samples before flotation, the formula for this calculation can be seen as Equation 5.

$$Copper\ Sulphide\ Recovery = \frac{Mass\ of\ Concentrate \times CuS\%_{Concentrate}}{Mass\ of\ Sample \times CuS\%_{Sample}} \qquad \textit{Equation 5}$$

Gangue acid consumption – Performed on all leached samples

There are three major classification of acid consumption that can be determined from the leach tests. Total Acid Consumption (TAC), Fresh Acid Consumption (FAC) and Gangue Acid Consumption (GAC), it is critical to understand what is defined by these terms and how they are calculated.

TAC – Defined as the total acid consumed by the sample, signifying the acid used to dissolve the target minerals as well as the gangue minerals. This value is expressed in kg of acid per dry metric ton of ore (kg_{acid}/MT_{ore}). TAC can be calculated from the experimental data using Equation 6:

$$TAC = \frac{V_{DA}C_{DA} - (V_{PLS}C_{PLS} + V_{WW}(C_{WW} - 0.085))}{M_{S_i}} = \frac{W_{Acid}}{M_{S_i}} \quad \textit{Equation 6}$$

The constant value of 0.085 is to account for the difference of acid between the wash water at a pH of 2.5 and the PLS at a pH of 1.8.

Gangue – Defined as any mineral that the mine does not process and/or sell as a product. In the case of the operation being investigated, all minerals other than copper or cobalt is considered gangue material.

GAC – Acid consumed by gangue minerals. GAC can be calculated using Equation 7:

$$GAC = \frac{W_{Acid} - \left(\left(\frac{MW_{H_2SO_4}}{MW_{Cu}} \times \Delta W_{Cu}\right) + \left(\frac{MW_{H_2SO_4}}{MW_{Co}} \times \Delta W_{Co}\right)\right)}{M_{S_i}} \quad \textit{Equation 7}$$

FAC – Defined as the fresh acid required to enter the plant for the purpose of leaching. The reason why it differs from the GAC is because the plant does not have a cobalt SX-EW circuit and does not regenerate any acid from the cobalt leached. In conjunction to that, there are 3 SX trains, 2 will be operated as High Grade (HG) PLS trains and 1 as a Low Grade (LG) PLS train. The acid from the LG train will go into the displacement wash circuit and won't be used in the leaching process. This means that

effectively only 2/3rds of the copper in solution will regenerate acid usable in the leaching circuit. Thus, FAC can be calculated using Equation 8:

$$FAC = \frac{W_{Acid} - \left(\frac{2}{3}\left(\frac{MW_{H_2SO_4}}{MW_{Cu}} \times \Delta W_{Cu}\right)\right)}{M_{S_i}} \qquad \textit{Equation 8}$$

The description for the symbols used in Equation 6, Equation 7 and Equation 8 can be seen in Table 8.

Table 8: List of symbols for acid consumption calculations

Symbol	Description	Unit
V_{DA}	Volume of Dilute Acid used	L
C_{DA}	Concentration of Dilute Acid	g/L
V_{PLS}	Volume of PLS	L
C_{PLS}	Free Acid in PLS	g/L
V_{WW}	Volume of Wash Water	L
C_{WW}	Free Acid in Wash Water	g/L
MW_{xxx}	Molar Weight of Element/Molecule	g/mol
ΔW	Weight of Element Extracted	g
M_{Si}	Weight of Sample before Leaching	kg
W_{Acid}	Weight of Acid Used for Leaching	g

Particle size analysis – done on all samples to verify grind size. Full PSDs were not performed on all samples but were done on certain samples and it will be specified in the text when is has been done.

Any samples greater than 500g was first split and reduced to a sample size of between 100 and 500g.

A RO-Tap RX-29 was used to determine the percentage solids passing the desired screen size, notably 75µm, 150µm and 212µm screens. A rubber washer was placed on each of the screen/s to prevent blinding of the sieves. The screen/s and a pan was/were placed in the sieve shaker and was shaked for 10 minutes. Any particle analysis required between 75µm and 38µm was done by wet screening

and all particles <38µm was submitted for analysis by the Malvern Particle size analyser.

Free acid analysis – Performed on all acidic solutions, particularly leachate and washate solutions from the acid leach tests. Free acid analysis was done by NaOH titration in the analytical laboratory on site by using a Mettler Toledo T5 auto-titrator.

Chemical analysis – Performed on all solid and liquid samples. Copper and cobalt analysis were done in the analytical laboratory using and Agilent flame atomic absorption spectrometer (AAS). Determining the contained copper and cobalt in liquid samples are simple and were directly diluted and used in the AAS. But when solid samples were submitted to the analytical laboratory two different procedures were followed to determine the total copper and cobalt in solids and the copper and cobalt present as oxides in the sample, otherwise known as acid soluble copper and cobalt. Taking the difference between these two results equalled what is called the acid insoluble copper and cobalt present in the sample, referred to in this thesis as copper and cobalt sulphides.

All solid samples submitted to the analytical laboratory was pulverised to a particle size of P_{100} 5µm. To determine the total copper and cobalt value in the sample 0,5g of the pulverised solids were digested in aqua regia on a hotplate underneath a fume hood. The digested solution was then diluted and sent for analysis by the AAS. To determine the copper and cobalt as copper oxide in the sample, the P_{100} 5µm sample was leached in concentrated sulphuric acid in Erlen Mayer flasks that were placed on a shaking table under a fume hood. The digested sample was then sent for analysis by AAS. Calcium, magnesium, iron and manganese were analysed using a Spectrolab Inductively Coupled Plasma – Optical Emission Spectrometer (ICP-OES).

Table 9: Summary of elements analysed on site and the method used for analysis,

	Element Analysed							
Method	TCu	CuOx	TCo	CoOx	Ca	Mg	Fe	Mn
AAS	X	X	X	X				
ICP					X	X	X	X

4 Results and discussion

4.1 Particle size tests

To determine the desired grind size for the acid leach plant, a screening was done on geological field samples, these samples were classified as OBI and OBS. To compare the acid consumption, the total acid consumption values are considered. From literature it was found that when you have a reduction in your particle size you are likely to get a higher recovery of copper and cobalt.

The copper oxides leach efficiency ($\%_{CuOx}$) is used for the comparison of the results as the sulphide fraction is not leachable under atmospheric acid leach conditions. The standard deviation in the leach results are also given as a percentage value, since standard deviation values are comparable to the values that the results are reported in. The leach test results can be seen in Table 10.

Table 10: Effect of particle grind size on the total acid consumption as well as the leach efficiency of copper oxides

OBI						
	TAC (kg/MT)			Leach efficiency ($\%_{CuOx}$)		
P_{80}	75µm	150µm	212µm	75µm	150µm	212µm
Average	152.6	140.2	140.2	91%	89%	89%
Std Deviation	3.56	2.50	4.54	0.4%	2.3%	2.0%
OBS						
	TAC (kg/MT)			Leach efficiency ($\%_{CuOx}$)		
P_{80}	75µm	150µm	212µm	75µm	150µm	212µm
Average	128.9	124.1	115.8	92%	91%	90%
Std Deviation	0.53	2.29	2.34	0.4%	0.7%	0.4%

From the initial screening it can immediately be seen that the acid consumption increases as the particle size decreases. The OBI sample's acid consumption and leach efficiency increased from 140.2 kg/MT to 152.6 kg/MT and from 89% to 91% for the 212µm and 75µm samples respectively. The OBS sample's acid consumption and leach efficiency increased from 115.8 kg/MT to 128.9 kg/MT and from 90% to 92% for the 212µm and 75µm samples respectively.

Notably, the acid consumption values changed more with respect to particle size than the leach efficiency did. The standard deviation also exhibited an interesting trend, as

the standard deviation decreased with the decrease in particle size. This could mean that predicting the behaviour in the plant environment could be more difficult since a smaller particle would yield a more repeatable behaviour.

To determine if the results were significant a single factor ANOVA test was performed to determine if the nul hypothesis can be rejected. The ANOVA analysis was only comparing the particle sizes with each other. This means that 2 ANOVA tests were run for the fresh acid consumption, one to compare the results from OBI and one set of results for OBS. The same was done for the copper oxide leach recovery.

All the tests were performed in pentaplicate, thus all between groups degrees of freedom was 2 and all within groups degrees of freedom was 12, which can be obtained by subtracting the within groups degrees of freedom from the total degrees of freedom which was 15. The ANOVA analysis results can be seen in Table 11.

To determine if the results vary statistically, a F value larger than the F-crit value was required, alternatively a P value smaller than 0.05 is required to reject the null hypotheses. From the ANOVA analysis it can be seen that 3 of the 4 result sets were found to be significant, the exception being the leach efficiency from the OBI ore. The OBI's leach efficiency could be due to the relatively high average cobalt content of 0.76%$_{TCo}$ compared to the 0.16%$_{TCo}$ of the OBS.

Table 11: Particle size effect ANOVA analysis

	OBS		OBI	
Statistical Parameter	**TAC (kg/MT)**	**Leach efficiency (%$_{CuOx}$)**	**TAC (kg/MT)**	**Leach efficiency (%$_{CuOx}$)**
F	16.2100	16.1068	15.0738	0.7766
F-crit	3.8853	3.8853	3.8853	3.8853
P-Value	0.0004	0.0004	0.0005	0.4818

To further analyse the results, a student's T-Test was used to compare the results from the 75µm tests to the results of the 150µm and 212µm test results, as well as comparing the 150µm results to the 212µm results. T-Test values of $p<0.05$ is required to reject the null hypothesis and determine if the results were significant. In the particle

size tests, the null hypothesis indicates that there is no statistical significance between the results obtained by decreasing the particle size. The results between the 150µm and 212µm particles were compared to the 75µm and 150µm tests resulting in a 2 by 2 matrix of results. The 150µm to 150µm comparison has no mathematical significance. The results from the statistical analysis can be seen in Table 12.

Table 12: Student T-Test results for comparing 75 µm, 150 µm and 212 µm results

OBI				
	TAC (kg/MT)		Leach efficiency (%Ox)	
P-values	75µm	150µm	75µm	150µm
150µm	**0.0001**	-	0.128	-
212µm	**0.011**	0.990	0.045	0.961
OBS				
	TAC (kg/MT)		Leach efficiency (%Ox)	
P-values	75µm	150µm	75µm	150µm
150µm	**0.0161**	-	**0.029**	-
212µm	**0.000**	**0.010**	**0.001**	0.490

When the acid consumption value results of the 75µm were compared to the 150µm and 212µm results all the T-Tests returned values of P<0.05, indicating that the results were significant. The results from the leach efficiency were not as conclusive as the acid consumption value results. For the OBI sample it was clear that there was no significant difference between the leach results, but the OBS sample suggested that the results were significant, and the null hypothesis should be rejected.

The cost of acid has a significant effect on the hydro refinery's profitability. To determine which particle size would be used for the remainder of the test campaign Equation 9 was used to compare the profitability of each of the particle sizes:

$$\frac{US\$}{h} = \dot{m}_{ore}\left((\%_{cu} \times Eff\%_{Cu} \times Price_{Cu}) - \left(\frac{TAC}{1000} \times P_{acid}\right)\right) \qquad \textit{Equation 9}$$

With ṁ being the feed rate to the FTAL plant, which was taken as 1450 t/h, based on the milling capacity of the existing mills. $\%_{Cu}$ is the percentage copper expected in the feed which was calculated as 3.45%. $Eff\%_{Cu}$ is the copper leach efficiency as obtained from the leach test results found in Table 10. The $Price_{Cu}$ is the price of

LME A grade copper which was US$5900 per ton at the time of calculation. TAC, being the total acid consumption as found in Table 10. P_{acid} is the price of acid per ton delivered to site, which was US$400 at the time of calculation. The results from the 150µm and 212µm were compared to the 75µm test that had the highest copper leach efficiency. The results can be seen in Table 13.

Table 13: Net profit comparison of different particle sizes

US$ per hour			
P_{80}	**75µm**	**150µm**	**212µm**
OBI	$ 180 076.23	$181 365.28	$181 365.28
OBS	$ 196 773.70	$196 606.23	$198 468.75
OBI Comparative Profit/Loss	-	$ 1 289.05	$ 1 289.05
OBS Comparative Profit/Loss	-	$ -167.48	$ 1 695.05
Total Weighted Profit/Loss	-	$ 117.07	$ 707.49

The weighted average profit/loss value was calculated by multiplying the individual OBI and OBS profit/loss values with the percentage of their contribution as found in Table 6, and adding them together. The 212µm is in fact the most profitable, but the factor that is not included in this calculation is the loss of recovery of sulphide minerals during the flotation process when a grind size of 150µm is exceeded. This does not form a central part of the study, but it was found in the initial tests that the copper sulphide recovery drops by 20.2% from 68.9% to 48.7% for 150µm and 212µm respectively and is discussed in more detail in the Percentage solids tests section. The loss in copper flotation would account for approximately a US$17 500 000-00 difference per annum. Thus, all remaining tests excluded comparisons of particles with a size of 75µm.

Further analysis show that the flotation recovery of the approximately 5-15% copper sulphides fraction of the total contained copper decreases on average from 55% to 42% for 150µm and 212µm respectively. The effects are much more pronounced for certain lithologies as can be seen from the results in Table 14.

Table 14: Flotation recovery of copper sulphide minerals present in predominantly copper oxide containing ore for 150µm and 212µm particles sizes

Lithology	Ore Body	Copper Sulfide %	150 µm Recovery	212µm Recovery
RSF	FNSR	0.26	51%	29%
RSF	OLIVEIRA	0.84	84%	90%
RSF	VIRGULE	0.89	47%	74%
RSF	VARIANTE	0.00	7%	13%
RSC	FNSR	0.23	96%	77%
RSC	OLIVEIRA	0.59	94%	38%
RSC	VIRGULE	0.03	47%	6%
RSC	VARIANTE	0.03	12%	16%
SDB	FNSR	0.01	81%	47%
SDB	OLIVEIRA	0.58	81%	75%
SDB	VIRGULE	0.65	28%	36%
SDB	VARIANTE	0.04	48%	40%
SDS	FNSR	0.22	52%	6%
SDS	OLIVEIRA	0.13	98%	71%
SDS	VIRGULE	0.04	2%	10%
SDS	VARIANTE	0.05	72%	40%
DSTRAT	FNSR	2.20	63%	72%
DSTRAT	OLIVEIRA	0.39	99%	85%
DSTRAT	VIRGULE	0.93	31%	38%
DSTRAT	VARIANTE	2.22	71%	50%
RATGR	FNSR	0.82	74%	84%
RATGR	OLIVEIRA	0.16	83%	76%
RATGR	VIRGULE	0.08	13%	9%
RATGR	VARIANTE	0.21	9%	10%
RATLILAS	FNSR	0.02	37%	14%
RATLILAS	OLIVEIRA	0.01	3%	11%
RATLILAS	VIRGULE	0.01	95%	11%
RATLILAS	VARIANTE	0.07	2%	13%

When the flotation results from the lithologies are grouped and averaged it was found that the largest flotation differences were for RSC, SDB, SDS and Ratlilas. Ratlillas however contributes a negligible amount of copper as received from the mine plan in Table 6. RSC, SDB and SDS however contribute 41%, 27% and 4% of the total ore from the mine to the refinery respectively. The averaged flotation results for the lithologies can be seen in Table 15.

Table 15: Flotation performance results of 150µm and 212µm particle sizes, grouped by lithology

Lithology	150 µm Recovery	212 µm Recovery	% Flotation Performance Difference (212 µm – 150 µm)
RSF	47%	51%	-8%
RSC	62%	34%	45%
SDB	60%	49%	17%
SDS	56%	32%	43%
DSTRAT	66%	61%	8%
RATGR	45%	45%	-1%
RATLILAS	34%	12%	65%

As a result, based on the gangue acid consumption, leach recovery as well as flotation recovery, the optimum grind selected for the subsequent tests were particles with a P_{80} of 150µm.

4.2 Percentage solids tests

Determining the effect of percentage solids on the leach efficiency as well as the acid consumption, the blended core samples as described in the Materials and material preparation section was used to leach the 150µm samples at 20%, 30% and 40% solids for a period of 4 hours at a pH of 1.5 in duplicate. From literature it is expected to find a higher leach efficiency with a decrease in solids concentration in the leach slurry. Tests done on the leaching of malachite with citric acid showed that above 5%$_{wt}$ had no significant change in the dissolution efficiency of the mineral (Shabani, et al., 2012). Increasing the solids concentration could lead to a situation where the reaction becomes diffusion layer limiting due to the high concentration of copper ions in the bulk solution. The results for the test results can be seen in Table 16.

Table 16: Effect of percentage solids on acid consumption as well as leach efficiency

% solids	20	30	40
TAC (kg/MT)	142.84	142.98	136.33
$\%_{CuOx}$ leached	97.6%	98.2%	97.0%
$\%_{CoOx}$ leached	93.1%	89.7%	95.3%
$\%_{CoTot}$ leached	84.9%	78.6%	82.9%
PLS Tenor (g/L)	8.66	12.05	16.00

The tests showed that there is an increase in copper leach recovery from 20%$_{wt}$ to 30%$_{wt}$ solids from 97.6% to 98.2% respectively, a decrease in leach recovery from 98.2% to 97.0% for the 30%$_{wt}$ and 40%$_{wt}$ solids samples respectively, these are all very high recoveries and could be due to experimental and analytical errors. The total acid consumption value follows a similar trend but is more pronounced in the high solids content test, the TAC values found for the 20%$_{wt}$, 30%$_{wt}$ and 40%$_{wt}$ solids tests were 142.84, 142.98 and 136.33 kg/MT respectively.

Diffusion is always driven by a concentration difference, where a molecule moves from an area of high concentration to an area of low concentration. As the percentage solids increase in the acidic leach slurry, the more copper is available to be leached in the closed system. With a reduced volume of solution, the concentration of dissolved metal in solution will increase. An increase in copper concentration in the bulk solution will decrease the diffusion rate of copper from the particle through the product layer that formed around the leached particle. The phenomenon could also be due to a reduced mixing efficiency caused by an increased viscosity of the higher solids content.

From the test results, it was decided that the optimal leach slurry would contain 30%$_{wt}$ solids or a slurry with a specific gravity of 1.24 kg/L if a solids density of 2.75kg/m^3 is assumed.

4.3 pH tests

The cost of sulfuric acid makes up 50% of the operational cost of a copper leach plant that runs an atmospheric acid leach process. Optimization of the acid usage is essential, the dissolution of copper oxides as a function of pH was thus tested to optimise leaching pH.

Determining the effect of pH on the leach efficiency as well as the acid consumption of the leaching process, the blended core samples as described in the Materials and material preparation section was used to leach the 150µm samples at 30% solids for a period of 4 hours at a pH of 1.0, 1.5 and 2.0. All the tests were done in duplicate.

The results for the leach tests at the various pH values can be found in Table 17. At the three pH values that was being tested, the mass loss was relatively constant for all three conditions. The difference in dissolution efficiency between the pH 1 and pH 1.5 was less than 0.1%. A slight drop in efficiency was only observed at a pH of 2 when the dissolution dropped from 95% to 93%. A remarkable difference was however found in the gangue acid consumption value that varied from 121.49kg/t_{ore} to 65.63kg/t_{ore} for pH 1 and 2 respectively.

Table 17: Effect on acid consumption as well as copper recovery due to pH variation

pH	1	1.5	2
% mass loss	10.7	10.3	9.6
TAC (kg/t)	182.17	142.98	124.77
GAC (kg/t)	121.49	80.98	65.63
$\%_{CuOx}$ leached	97%	98.2%	93%
$\%_{CoOx}$ leached	93%	89.7%	72%

The sharp drop in the cobalt leach recovery can also be expected as cobalt will precipitate out at solution Eh values below -0.25V and pH values greater than 1.0. This phenomenon can clearly be seen in the phase diagram for copper and cobalt in the presence of iron Figure 21 (Seo, et al., 2013).

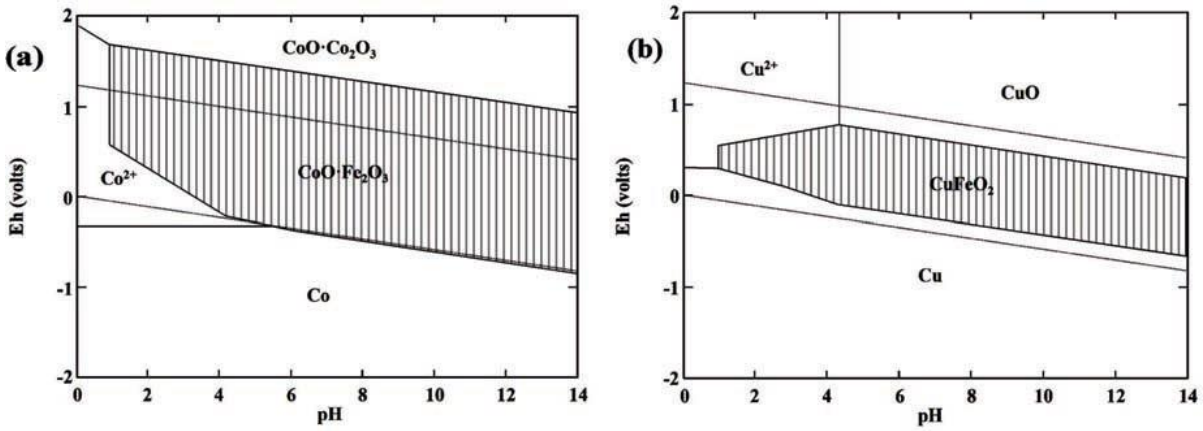


Figure 21: Phase diagrams for copper (b) and cobalt (a) in the presence of Fe (Seo, et al., 2013)

4.4 Temperature tests

Determining the effect of temperature on the leach efficiency as well as the acid consumption of the leaching process, the blended core samples as described in the Materials and material preparation section was used to leach the P_{80} 150µm samples at 30% solids for a period of 4 hours at a pH of 1.5 at 30°C, 45°C and 60°C All the tests were done in duplicate. The results for the leach tests at the various temperatures can be found in Table 18.

Table 18: Effect on acid consumption as well as copper recovery due to temperature variation

Test Condion Variable	30°C	45°C	60°C
$\%_{CuOx}$ leached	98.2%	98.2%	98.5%
$\%_{CoOx}$ leached	89.7%	88.0%	88.8%
$\%_{CoTot}$ leached	78.6%	87.1%	88.4%
TAC (kg/MT)	143	144	147

An assumption derived from the Arrhenius equation states that an increase in temperature generally leads to an increased leach efficiency. The effect of temperature however did not seem to have a great effect on the recovery of copper, and the acid consumption values obtained from the tests are inconclusive. As seen with the pH tests, the cobalt seems to be much more affected by the change in temperature. The change in cobalt recovery could be due to observed pH in the system caused by the change in temperature. When temperature increases in a system where pH is being measure two things occur:

1. The H^+ atoms inside of the system vibrate more due to the increased temperature and as a result the observable concentration of hydrogen ions in solution increases.

2. Increased temperatures also increase the ability of water to ionise and produce more hydrogen ions in solution, as displayed in Reaction 9.

$$H_2O\ (l) \leftrightharpoons H^+(aq) + OH^-(aq)$$ *Reaction 9*

This effect gets larger as the pH increases, i.e. basic solutions are affected by it a lot more than acidic solutions (Westlab Canda, 2017). The Hanna HI 2211 pH meters include automatic temperature compensation, but the effect might just have been large enough to get close to a pH value in the phase diagram where cobalt no longer exist as a metal ion in solution.

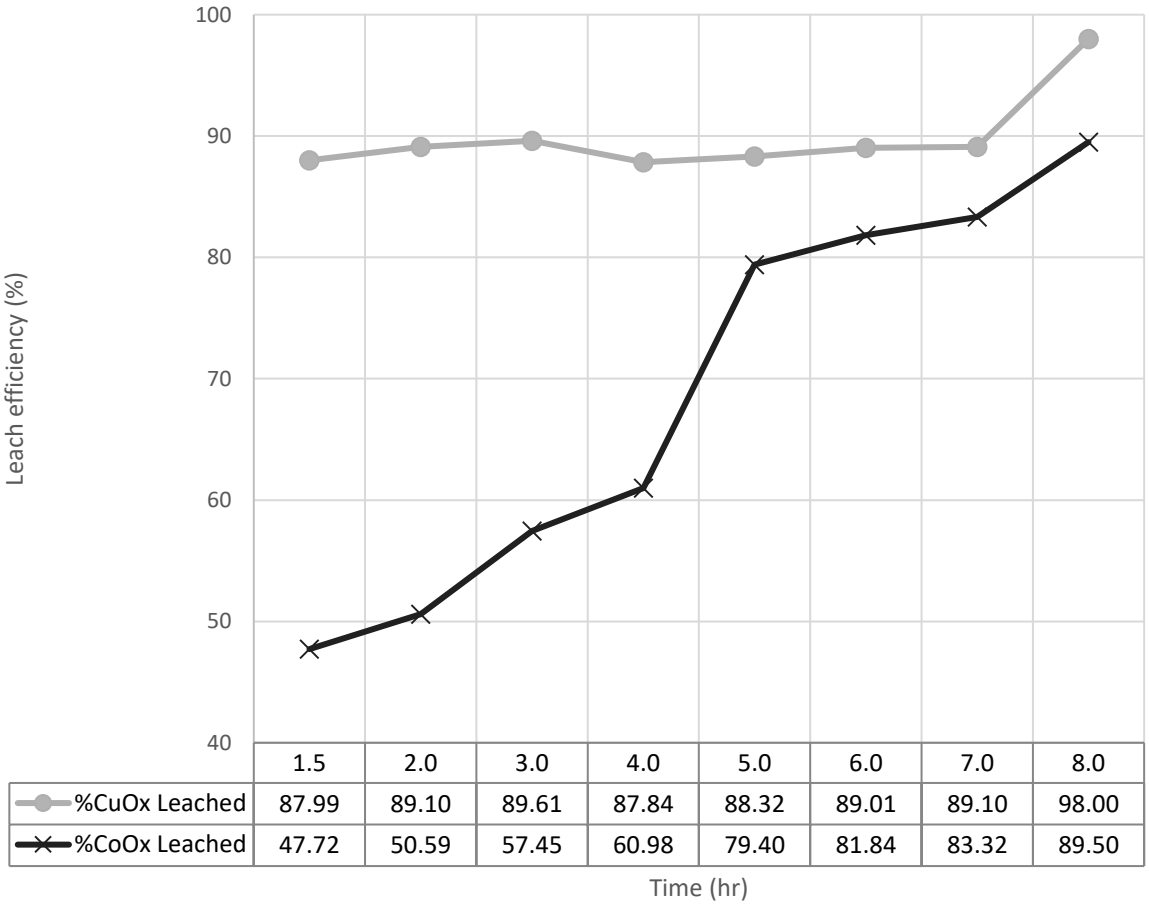
A similar phenomenon has been seen for the measurement of Eh of solutions, where the measured Eh of a solution decreases with an increase of temperature (Rodkey, 1959). Water exhibited a decrease in Eh value from 251mV to 238mV for 10°C and 30°C respectively (James, et al., 2004).

4.5 Leaching Kinetics Tests

The leaching kinetics tests are divided into two separate test campaigns. The first set of kinetic test were done on the blended sample as used for the tests in Particle size tests, Percentage solids tests, pH tests and Temperature tests. These tests were pH as well as Eh controlled. As mentioned in the Materials and material preparation section, a large amount of core samples became available after the initial screening was completed. The core samples exhibited a large variation in their acid consumption values and to a smaller extent their copper leach efficiencies. To analyse the effects 4 samples were earmarked to analyse their leach kinetics.

4.5.1 Leach kinetics on blended sample

This sample was mainly used to determine the size of the agitated leach reactors required for adequate recovery of cobalt. As it's known from literature and previous experience that copper oxides requires 1 to 2 hours of residence time for adequate leaching of 90-95%. On the other hand, the dissolution of cobalt requires 4-6 hours of residence time for adequate leaching due to the redox reaction required to leach the cobalt from the mineral heterogenite, where cobalt is found in its 3+ oxidation state. As the objective of this leach test was to determine the residence time required to leach cobalt sufficiently, the first sample was taken at 90 minutes, followed by 120 minutes and then every hour after that.



	1.5	2.0	3.0	4.0	5.0	6.0	7.0	8.0
%CuOx Leached	87.99	89.10	89.61	87.84	88.32	89.01	89.10	98.00
%CoOx Leached	47.72	50.59	57.45	60.98	79.40	81.84	83.32	89.50

Figure 22: Leach kinetics of copper and cobalt oxides from blended ore sample

The copper leach recovery was found to be stable for the whole duration of the test, a possible explanation for the sharp incline in recovery of the copper for the final sample in the kinetics test could be due to the method of sampling.

As the 8-hour sample is taken after the bulk sample has been leached, filtered and dried. Thus, a more representative sample is submitted as the final sample of the leach test. The value of 98.0% also corresponds better to the base case leach test that had a copper oxide recovery of 97.8%.

The cobalt leach recovery showed a very distinct, linear leach recovery from 1.5 to 4 hours from 47.7% to 68.9%. The sudden increase in cobalt leach recovery from 4 to 5 hours is due to the leach test only being Eh controlled by the addition of SMBS from 4 hours onward. Only after 5 hours does the trend even out, and as with the copper leach curve, there is a sudden increase in the final leach recovery value and could also be due to the method of sampling. This indicated clearly that the agitated leach tanks will need to be designed to the rate of reaction of the cobalt dissolution reaction, rather than that of the copper dissolution reactions.

4.5.2 Leach kinetics on core samples

The samples selected to be analysed for their copper kinetics can be seen in Table 19. The samples in Table 19 show variations in acid consumption figures, contained copper content, copper oxide recovery as well as cobalt leach recovery. It should be noted that the Eh was not controlled during these leach tests as the aim was to determine the leach kinetics of copper, rather than cobalt.

Table 19: Copper leach kinetics sample selection sheet

Fragment	Lithology	Copper Oxides in Feed (%)	Total Cobalt in Feed (%)	Copper Oxide Leach Recovery (%)	Cobalt Leach Recovery (%)	GAC (kg/MT)
FNSR	SDS	3.14	3.00	87.5	100.0	27
FNSR	SDB	5.95	1.57	95.2	12.7	15
OLIVEIRA	SDB	5.86	1.42	95.6	23.6	62
VIRGULE	RATGRIS	3.34	0.21	98.0	25.6	7

Although the focus was placed on the kinetics of copper, it can clearly be observed that the copper oxide recovery is directly correlated to the contained cobalt in the feed sample. This observation is further discussed in the Characterisation of leach performance of different lithologies section.

The recovery-time curves can be seen in Figure 23. From the curves it can be seen that the extraction of copper slows down significantly after 30 minutes, and practically stops after 60minutes, similar findings were made by Ntengwe (2010). Even when malachite was leached using citric acid, 89% of the copper was leached within the first 30 minutes, with a final leach recovery of 92% after 120 minutes (Shabani, et al., 2012). Thus, to determine the reaction rate constant only the first 30 minutes will be considered.

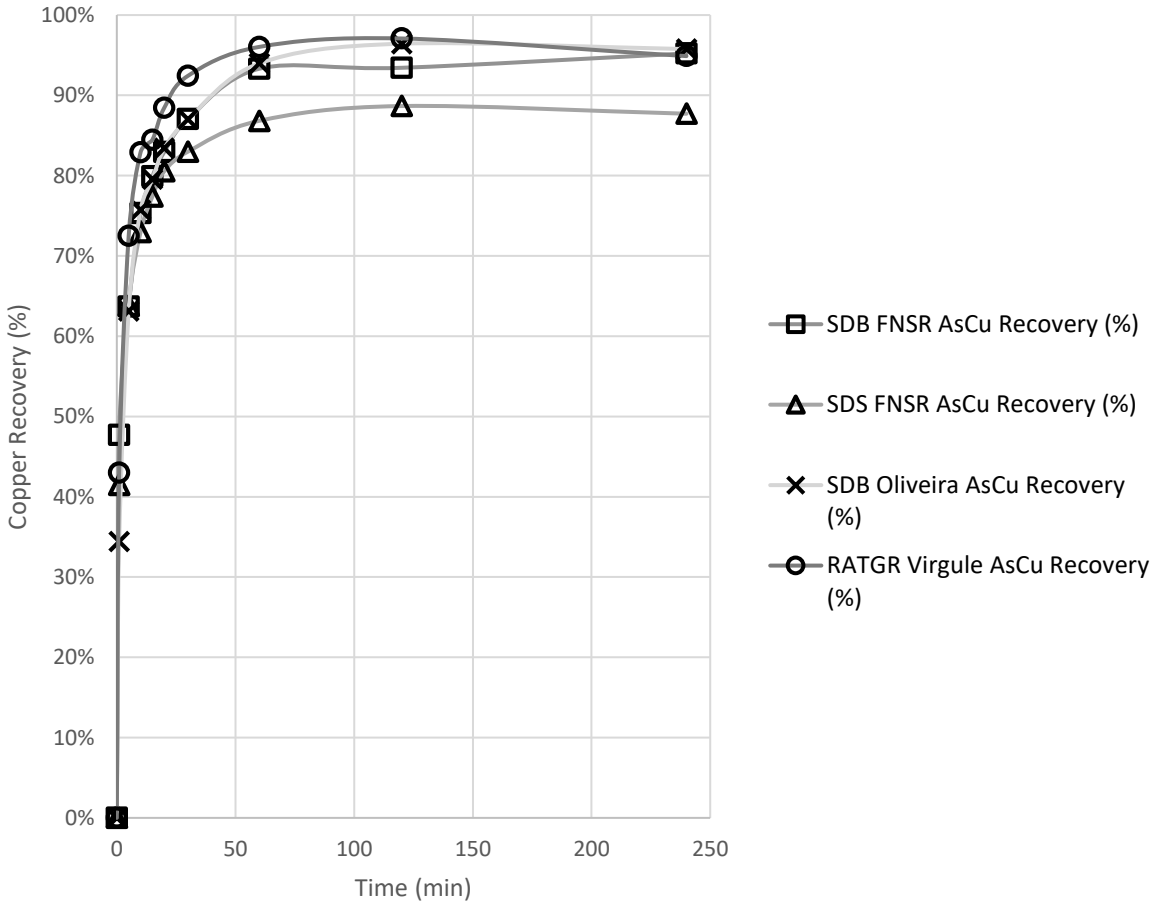


Figure 23: Copper oxide recovery of various lithology samples plotted against time

The dissolution of copper can be explained by the shrinking core model. Therefore, the diffusion and the surface reaction control models were investigated. If the reaction is controlled by product layer diffusion Equation 10 may be used:

$$k_d t = 1 - 3(1 - X_{Cu})^{\frac{2}{3}} + 2(1 - X_{Cu})$$ *Equation 10*

If the reaction is controlled by the surface reaction model Equation 11 may be used:

$$k_s t = 1 - (1 - X_{Cu})^{1/3} \qquad \textit{Equation 11}$$

With X_{Cu} being the fraction of copper leached, t is the reaction time and k_d and k_s are the rate constants. Equation 10 and Equation 11 were applied to the experimental results and the correlation coefficients for each sample were calculated the results can be seen in Table 20.

Table 20: k_d, and k_s values and correlation coefficients for different samples

Fragment	Lithology	Apparent rate constant (min^{-1}) for Copper		Correlation coefficient (R^2) for the fitted data	
		k_d	k_s	k_d	k_s
FNSR	**SDS**	0.0177	0.0199	0.918	0.854
FNSR	**SDB**	0.0199	0.0214	0.934	0.864
OLIVEIRA	**SDB**	0.0200	0.0214	0.941	0.883
VIRGULE	**RATGRIS**	0.0249	0.0248	0.929	0.873

These results indicate that the rate limiting factor for the leaching of copper from all the tested lithologies is product layer diffusion controlled. The rate constants and correlation coefficient were also calculated by fitting the experimental data to the film diffusion layer limiting model. The plotted datapoints had a logarithmic curve and the correlation coefficients found were 0.80, 0.80, 0.83 and 0.80 for SDS FNSR, SDB FNSR, SDB Oliveira and RATGRIS Virgule respectively, thus the model was not considered.

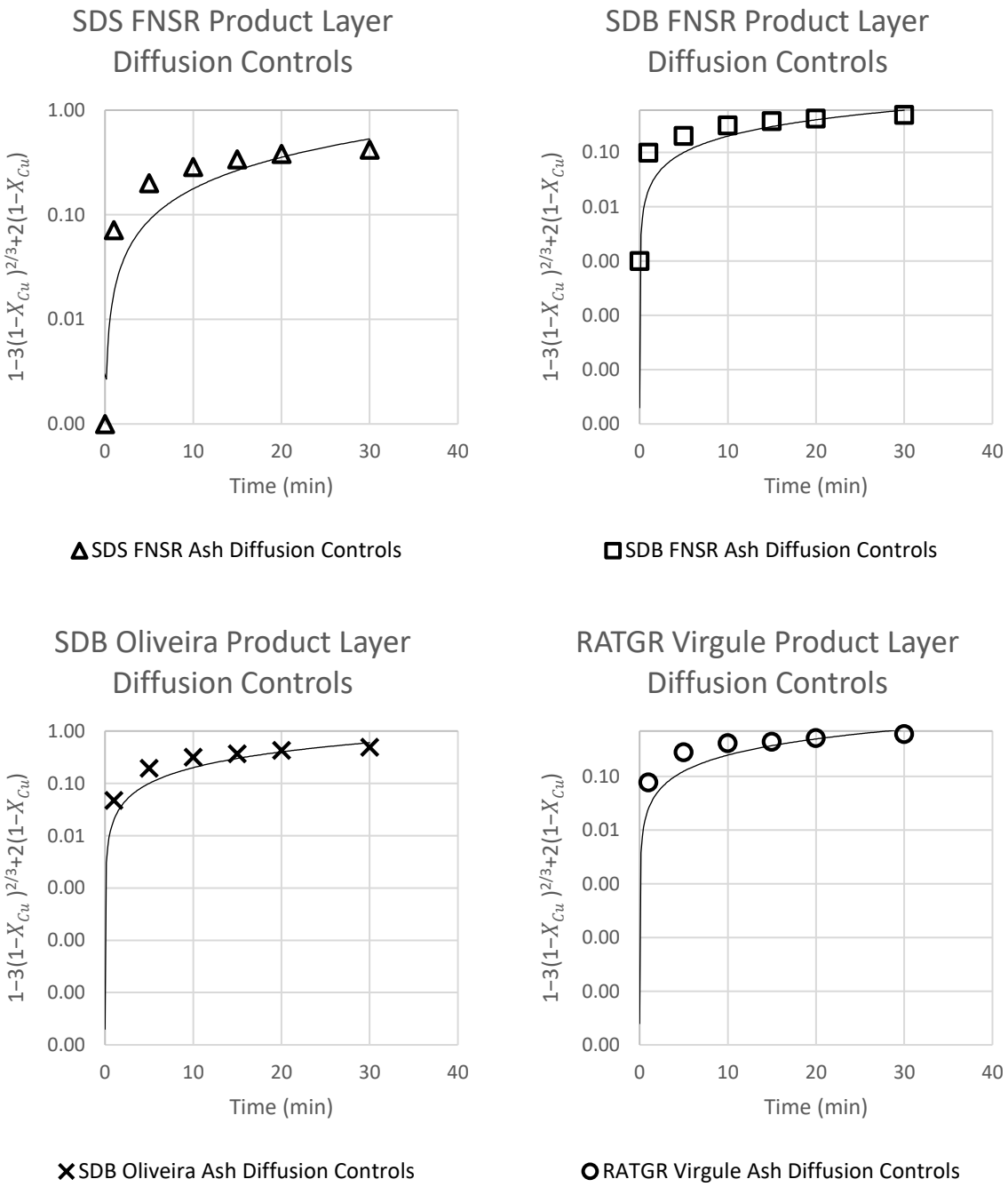


Figure 24: Plot of Equation 10 vs. the first 30 minutes of leach time for different lithologies

Even though the product diffusion layer limiting model best fits the leaching kinetics results, the deviation from the model indicates that other factors play a role in the leaching process. The development of a mathematical model that takes multiple rate limiting steps into account might be suitable for the leaching of the ore. Similar findings

has been made by Wanta, et al. (2018) whilst investigating atmospheric acid leach of nickel laterites using citric acid.

Another study by Safari, et al. (2009) found that the formation of a gel layer formed by the reaction between sulfuric acid and silica in hemimorphite, a zinc containing mineral, that formed dicilicate acid that covered the particle in a soft gel-like substance. A model that described the reaction as being a shrinking core – shrinking particle limited. A similar phenomenon could occur in the leaching of the ore in this study as a major gaunge constituent has been found to be quartz.

4.6 Characterisation of leach performance of different lithologies

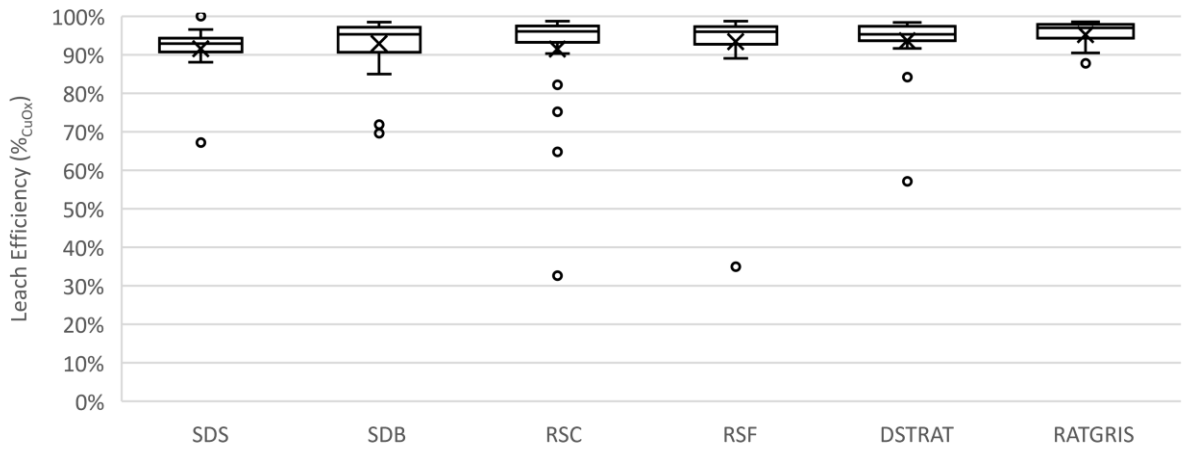
From the drilling campaign, 32 more core samples were tested to determine the leaching characteristics from the ore body. From the 32 cores, a total of 208 testable samples were floated and the flotation tails were leached in duplicate. Table 21 gives a breakdown of the number of samples floated and leached during the campaign.

Table 21: Test campaign leached samples summary

Lithology	Samples from KOV
SDS	20
BOMZ	21
SDB	38
RSC	35
RSF	33
DSTRAT	29
RATGRIS	23
RATL	9
Total	208

To profile the ore body, it is required to determine the expected oxidised copper leach recovery from each ore body as well as the gangue acid consumption from each lithology, the validity of the data was analysed statistically. The results for the leach efficiency as varied across the different lithologies can be seen in Figure 25. The box

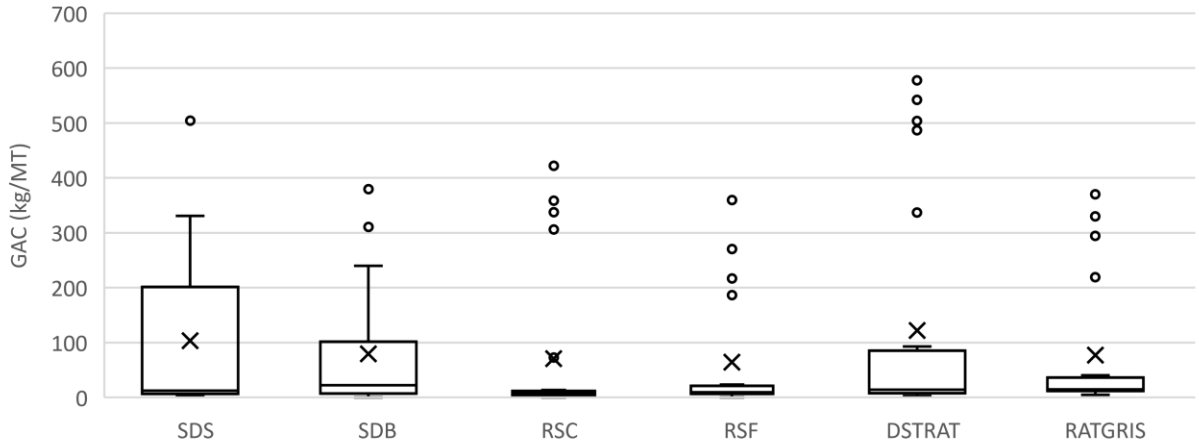
and whisker diagrams showed that the leach results had a narrow standard deviation and the average recovery will be a good indication of the expected recovery.



	SDS	*SDB*	*RSC*	*RSF*	*DSTRAT*	*RATGRIS*
Mean	92%	93%	91%	90%	94%	95%
Standard Error	0.016	0.011	0.024	0.036	0.016	0.009
Median	93%	95%	96%	96%	95%	97%
Standard Deviation	7%	7%	13%	20%	8%	4%
Count	18	35	30	31	26	19
Confidence Level(95.0%)	3%	2%	5%	7%	3%	2%

Figure 25: Leach efficiencies for SDS, SDB, RSC, RSF, DSTRAT and RATGRIS including statistical analysis

The same analysis was performed for the gangue acid consumption values obtained from the tests. The results can be seen in Figure 26. The GAC values for the lithologies showed a lot more variability, and the results included more statistical outliers. However, the main lithologies, SDB, RSC and RSF contribute to 82% of the ore bodies' minable ore, of which RSC makes up 50.2%. The main lithologies showed a more repeatable behaviour with regards to acid consumption. Predicting the GAC value solely on the geological characterisation is clearly not a solution that is robust, or reliable. It merely serves as a rough approximation of what to expect from a certain lithology.



	SDS	*SDB*	*RSC*	*RSF*	*DSTRAT*	*RATGRIS*
Mean	103.7	75.6	80.2	72.9	122.1	77.1
Standard Error	36.3	17.6	26.3	22.2	38.7	28.3
Median	12.2	19.4	8.4	10.8	14.3	15.0
Standard Deviation	153.9	102.7	144.3	121.6	197.4	123.3
Count	18	34	30	30	26	19
Confidence Level (95.0%)	76.5	35.8	53.9	45.4	79.7	59.4

Figure 26: Gangue acid consumption for SDS, SDB, RSC, RSF, DSTRAT and RATGRIS including statistical analysis

An easy and reliable method to predict the consumption of acid from exploration drill cores, geological grab samples as well as bi-hourly mill discharge samples is essential to avoiding unforeseen spikes in acid consumption at the refinery. The gangue acid consumption for each test was calculated using Equation 12. The following correlation between the gangue acid consumption and the calcium content was established, the results of each lithology can be seen in Figure 27.

$$GAC\left(\frac{kg_{H_2SO_4}}{DMT_{ore}}\right) = 44.667 \times Ca\%_{wt} - 1.0863$$ *Equation 12*

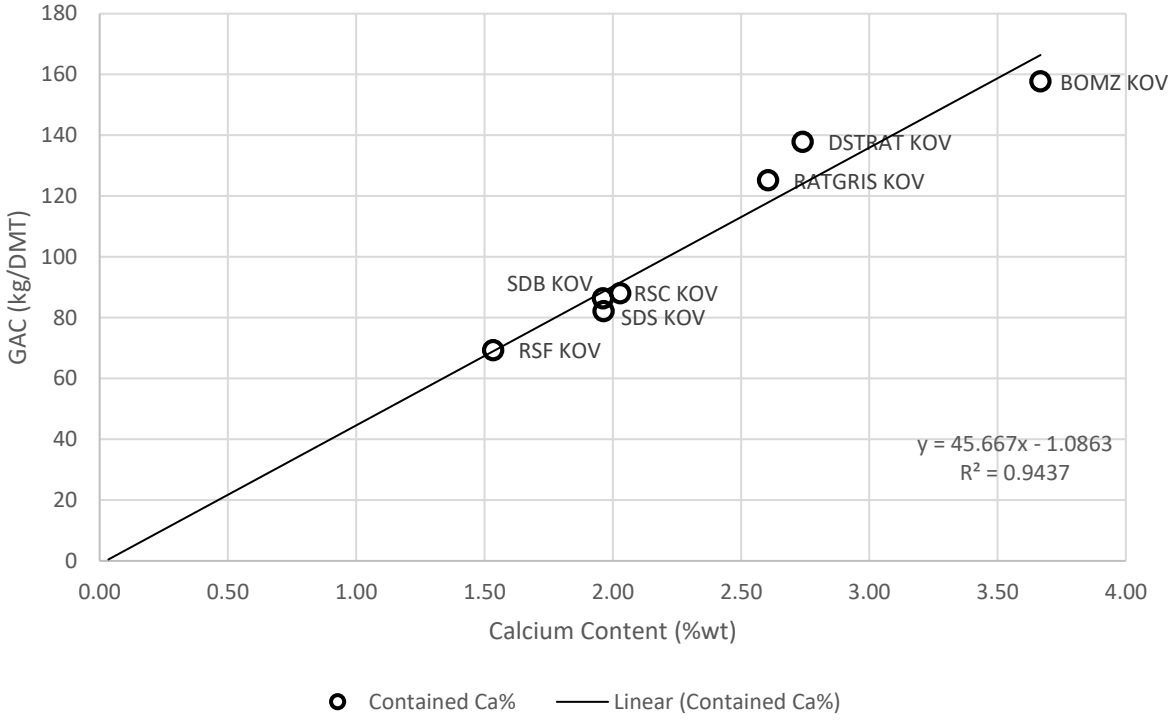


Figure 27: Gangue acid consumption as a function of contained calcium content

It would be expected that a similar trend exists for magnesium, and it was commonly assumed that the major gangue constituent was dolomite. Dolomite has the following chemical formula:

$$CaMg(CO_3)_2$$

The molecular weights of calcium and magnesium are 40.078g.mol^{-1} and 24.305g.mol^{-1} respectively. By dividing the molecular weight of calcium by the molecular weight of magnesium a molecular weight ratio of Ca:Mg in dolomite of ≈1.65 is calculated. When the contained calcium was plotted against the magnesium content two very distinct value sets could be identified, as seen in Figure 28. A regression line was drawn through the *Ca Series*, see Equation 13. This graph indicates that there are minerals containing Mg, that are not dolomite. And from Figure 29 it can be seen that there are minerals containing Mg that do not contribute to GAC.

$$\%Ca = 1.5718 \times \%Mg$$ *Equation 13*

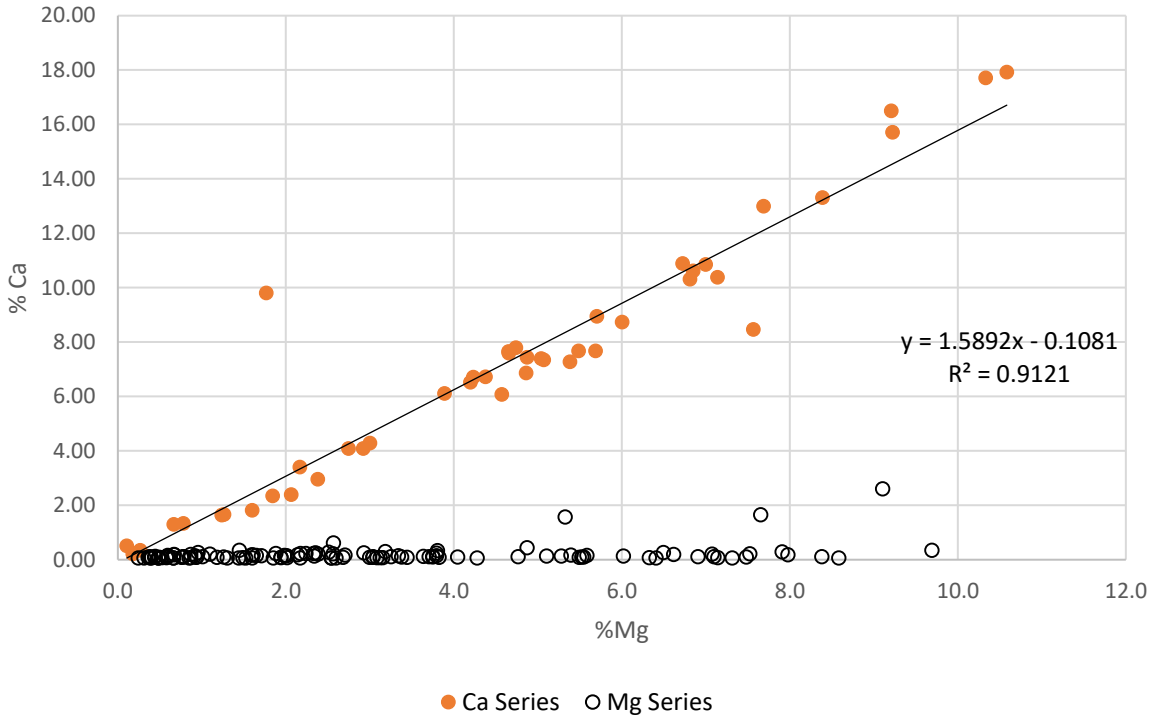


Figure 28: Contained calcium and magnesium

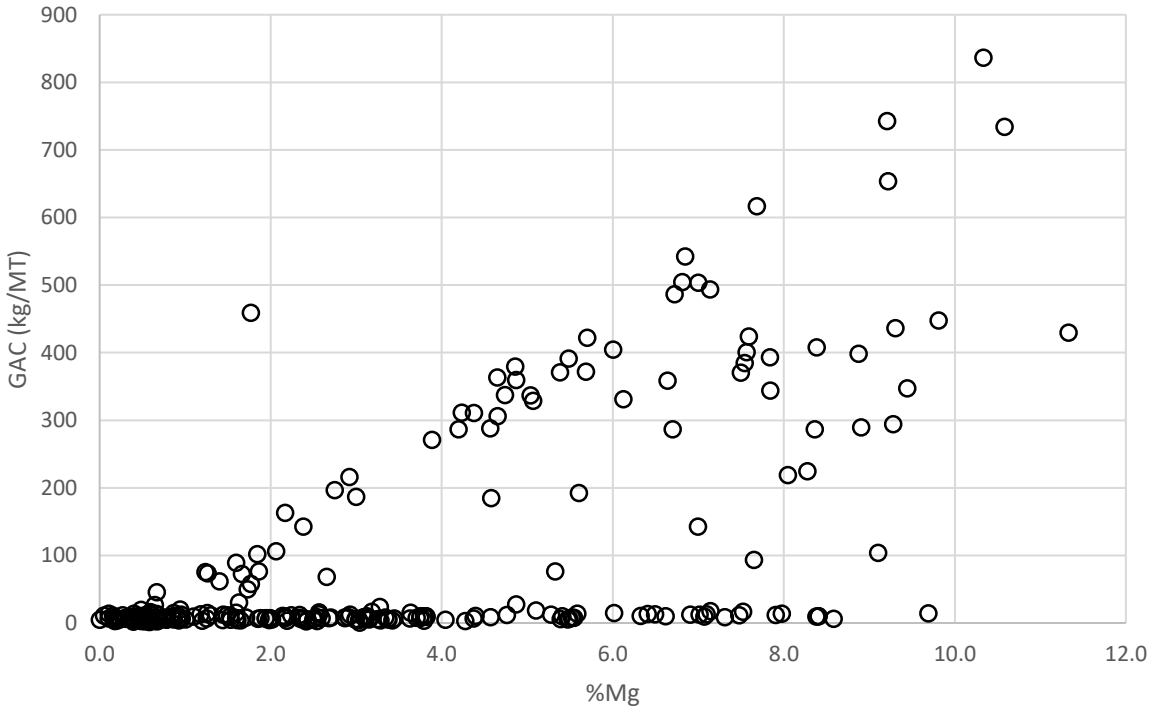


Figure 29: Gangue acid consumption versus contained magnesium

If the empirical value of dolomite ≈1.65 and the slope of the Equation 13 ≈1.57, are compared, the broad view that the major gangue constituent is in fact dolomite can be putative. Three samples were submitted for mineralogical analysis by QEMSCAN. The sample selection was based on acid consumption and recovery. The results for the pre-float tails can be seen in Table 22. The minerology confirms that the major carbonaceous mineral constituent is dolomite.

Table 22: QEMSCAN results for pre-float tails

Mineral Group	Mineral	KOVO755-SDS-FNSR-PFT2	KOVO640-SDB-OLIVEIRA-PFT1	KOVCH015-RSC-FNSR-PFT1
	Leach CuOx Recovery	88%	95%	99%
	Leach TCo Recovery	99%	46%	31%
	GAC (kg/DMT)	27.4	310.5	4.3
Cu Sulphides	Bornite	0.02	0.09	0.03
	Chalcopyrite	0.04	0.03	0.01
	Chalcocite	0.02	0.57	0.11
	Covellite	0.00	0.21	0.01
	Carrollite	0.00	0.14	0.04
Cu Oxides	Malachite	8.39	6.86	9.57
	Pseudomalachite	0.00	0.00	0.29
	Chrysocolla	0.06	2.01	0.18
Co Minerals	Asbolan	0.03	0.03	0.75
	Heterogenite-3R	10.11	1.74	0.95
Fe Sulphides	Pyrrhotite	0.04	0.01	0.01
	Pyrite	0.00	0.00	0.01
FeOxide	FeOxide	0.24	1.18	0.58
Quartz & Fledspar	Quartz	44.27	41.62	80.32
	Albite	0.01	0.01	0.01
	Anorthoclase	0.12	2.86	0.11
	Orthoclase	0.15	1.28	0.03
	Hastingsite	0.03	0.31	0.03
	Kyanite	0.04	0.06	0.05
Talc & Pyroxene	Talc_Enstatite	0.17	0.05	0.15
	Diopside	0.02	0.68	0.01
	Ferrosilite	0.45	0.30	0.26
Mica	Biotite	11.49	1.25	2.39
	Muscovite	11.95	9.90	1.13
Chlorites	Clinochlore	10.02	1.83	2.61
Carbonates	Ankerite	0.00	0.01	0.00
	Dolomite	0.76	25.87	0.07

Mineral Group	Mineral	KOVO755-SDS-FNSR-PFT2	KOVO640-SDB-OLIVEIRA-PFT1	KOVCH015-RSC-FNSR-PFT1
	Calcite	0.14	0.07	0.08
Gypsum	Gypsum	0.00	0.00	0.02
Other	Chromite	0.05	0.02	0.05
	Periclase	0.00	0.01	0.01
	Apatite	0.20	0.21	0.01
	Rutile	0.89	0.61	0.11
	Unknown	0.28	0.20	0.04
	Total	100.00	100.00	100.00

Mineralogical testing was done on three core samples, which were identified based on the following criteria:

1. The total copper grade had to be between 3.5% and 4.5%, to represent the expected feed grade to the acid leach plant.
2. The section of the core sample had to be from one of the three largest sections of the ore body, namely SDS, SDB and RSC.
3. A range of copper oxide recoveries had to be represented by the samples, one below, one above and one approximately the same as the expected leach efficiency of ±93%.
4. The samples had to represent a range of gangue acid consumption values.

To investigate isothermal, atmospheric acid dissolution behaviour of copper oxide minerals, notably malachite and chrysocolla which are hosted in three prominent gangue minerals specifically quartz & clays namely biotite and muscovite, the mineralogy had to be examined to explain the difference in recoveries and acid consumptions.

The three samples under consideration in Table 22 have a few distinct differences both mineralogically as well as their leach behaviour. The recovery was found to be 88%, 95% and 99% for the SDS, SDB and RSC samples respectively. Interestingly the cobalt content, seems to have a direct impact on the leach efficiency, as seen in the Leach kinetics on core samples section. Figure 30 plots the copper oxide recovery versus the contained cobalt content as analysed by the QEMSCAN. This could be due

to galvanic exchange, or redox reaction, occurring in the system, much like the common copper recovery process known as cementation, usually facilitated by the use of iron filings, following chemical Reaction 10.

$$Cu^{2+}(aq) + Fe(s) \rightarrow Fe^{2+}(aq) + Cu(s)$$ *Reaction 10*

The electrode potential for iron an heterogenite in a hydroxide system are -0.44V (Caroli & Sharme, 1978) and -0.563V (Behl & Toni, 1971). This would mean that copper in solution would precipitate out as cobalt is being oxidised.

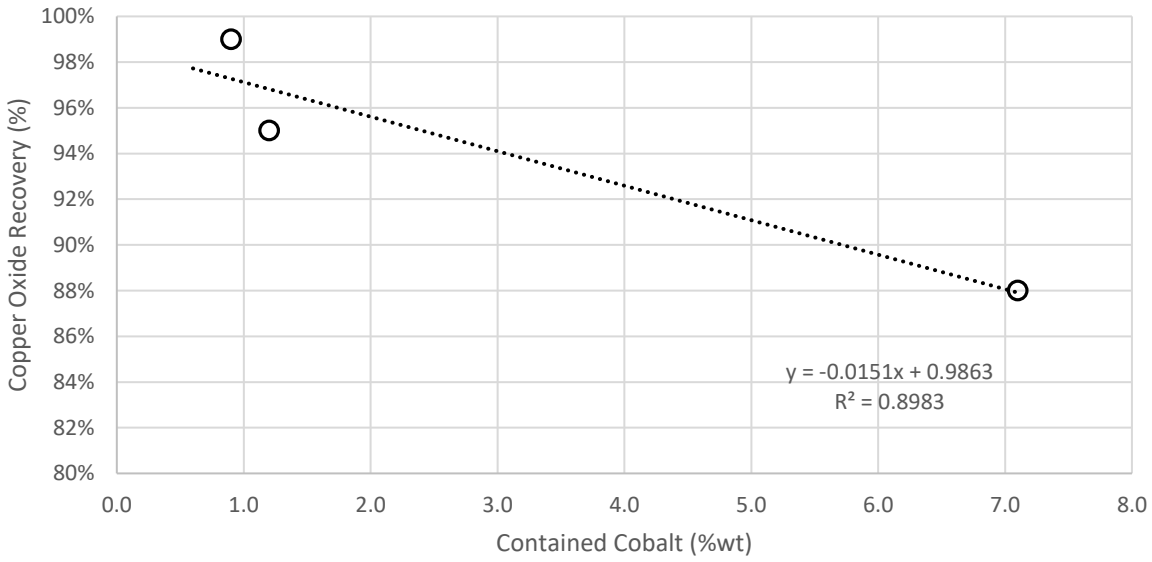


Figure 30: Copper oxide efficiency versus contained cobalt in feed, from mineralogical analysis

When the copper oxide recovery was plotted versus the contained cobalt in the feed, as obtained from the other core samples, a similar trend could be seen, and is illustrated in Figure 31. The R^2 value is low but a clear linear trend with a negative slope can be observed.

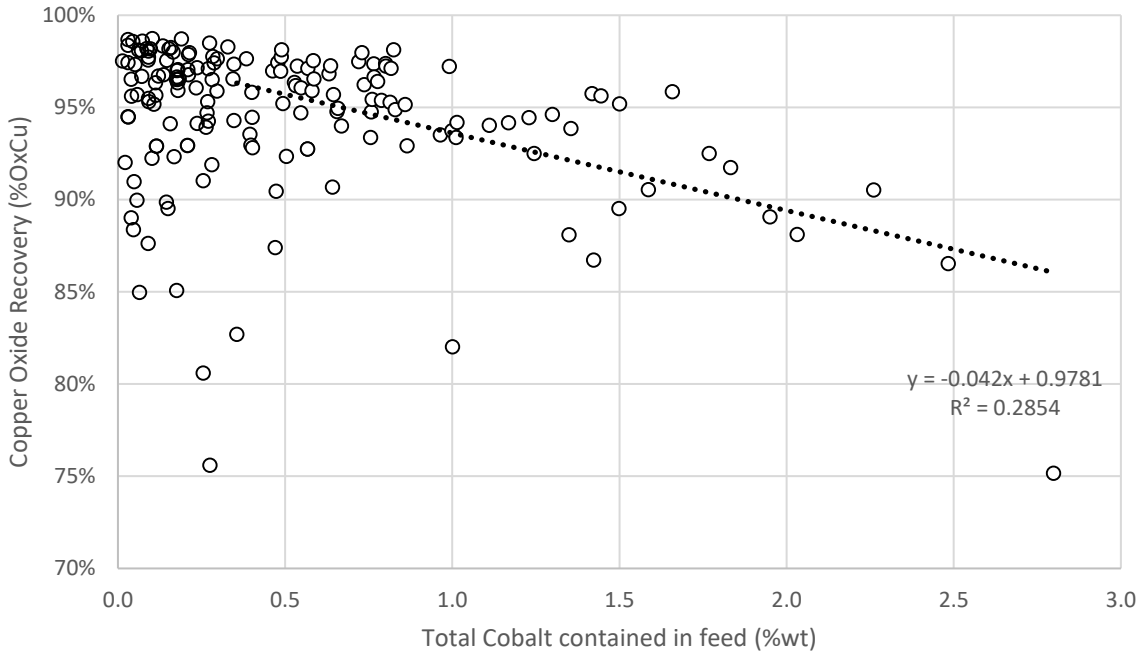


Figure 31: Copper oxide efficiency versus contained cobalt in feed, from core samples

The GAC values was found to be 27.4, 310.5 and 4.3 kg_{acid}/t_{ore} for SDS, SDB and RSC samples respectively. The major copper oxide bearing mineral conversely remained the same for all three samples. Malachite accounted for 99%, 77% and 95% of the copper oxide bearing minerals in the SDS, SDB and RSC samples respectively.

The mineralogical differences found in the host minerals can explain why the acid consumptions as well as leach performance variances are observed. It is very clear that the RSC sample had excellent copper leach recovery. From the mineralogy it was found that the minerals present were almost exclusively malachite and quartz, accounting for 9.57% and 80.32% of the minerals in the sample respectively. Quartz has a brittle tenacity, and displays conchoidal fracturing, resulting in fine powdery or crumby shards when subjected to milling or crushing. As a result, the copper oxide containing mineral, malachite, is very liberated, resulting in very high copper oxide recoveries.

The SDS and SDB samples have a more complex mineralogy. The SDS consists of 8.39% malachite, hosted in 44.27% quartz, 11.49% biotite, 11.95% muscovite, 10.02% clinchlore and 10.11% of the cobalt containing mineral hetrogenite.

The SDB sample also contained a lot of clays but has distinctly more calcium than any of the other two samples. Resulting in the high acid consumption value of 310.5 kg_{acid}/MT_{ore}. The major copper oxide constituents of the SDB sample are 6.86% malachite but also 2.01% chrysocolla hosted in 41.62% quartz, 9.90% muscovite and 25.87% dolomite.

The mica which is present in the SDS and SDB samples has sectile and elastic tenacity which means that the clays do not shatter when put under stress. During the crushing and milling of these minerals, they will bend, and will return to their original positions when the stress is released. Thus, rather than liberate or expose the copper and cobalt minerals they can laminate them in clay layers that first need to be exfoliated before effective leaching can commence. SDB shows a much better leach efficiency than SDS. This is due to the large quantity of dolomite instead of biotite. The dolomite reacts and dissolves easily under the leaching conditions and quickly exposes the copper oxides to the lixiviant. The Ca is indicative of carbonates, which consume acid.

5 Refinery mass balance

To quantify the effect that the mineralogy can have on the possible production of copper metal as plated copper cathodes and cobalt as a cobalt hydroxide cake, the average, minimum and maximum values found by the study was examined. The aim of the mass balance is to emphasise the importance of a proactive blending strategy, to minimise disturbances in feed composition. It also emphasizes the impact of not controlling the dissolved iron, ferric and ferrous, in the raffinate in conjunction with the addition of SMBS or SO_2 to the agitated leach tanks to ensure successful leaching of cobalt.

The following assumptions were made while setting up the mass balance:

1. The dry solids feed rate to the refinery based on the capacity of the mine was to be limited to 1450 tons of dry ore per hour. This feed is split between the two trains. i.e. 725 t/h/train.
2. The contained copper oxide minerals in the feed was to remain constant at 3.45%. This was determined by taking weighted average copper oxide copper results from the core samples as described in Materials and material preparation section as shown in Table 6.
3. Using the same method as for the calculation of copper oxides, a total cobalt feed grade of 0.55% would be used.
4. A washing efficiency of 92.2% in the counter current decantation section was used based on operational data, this washing efficiency was achieved by a wash ratio of 1.2.
5. Solvent extraction efficiency was to remain constant at 95.3% based on operational data.
6. The electro winning circuit was estimated to be 99% efficiency.
7. Cobalt recovery by selective precipitation was assumed to remain constant at 75%, based on historic plant data.
8. The plant availability was estimated at 90%.
9. Lime consumption for selective precipitation as well as tailings neutralisation was kept constant at 204kg per ton of ore fed to the plant, based on plant data.

10. It was assumed that the maximum copper recovery would be associated with the minimum cobalt recovery and vice versa.

Three scenarios investigated are:

Scenario 1: The averages from the data collected from the tests as described in the Characterisation of leach performance of different lithologies section was used to determine how much copper and cobalt can be expected to be produced per annum.

Scenario 2: The maximum copper and minimum cobalt recovery was assumed to be associated based on the results from the Characterisation of leach performance of different lithologies section.

Scenario 3: The minimum copper and maximum cobalt recovery was assumed to be associated based on the results from the Characterisation of leach performance of different lithologies section.

The results for the three scenarios can be seen in Table 23.

Table 23: Mass balance results based on experimental results

	Copper Produced KTPA	Cobalt Produced KTPA
Scenario 1 – Base Case	316	16.5
Scenario 2	339	2.8
Scenario 3	296	35.3

Furthermore a mass balance including a copper lock-up for the whole refinery was done to determine the copper inventory that can be expected with a target copper oxide feed grade of 3.45% copper with the typical mineralogy as found by the QEMSCAN analysis. The average recoveries from the test campaign were used in a steady state Metsim model, reactions typically found and discussed in the literature study were used in the leach reactors. Appendix B: Refinery Copper Balance breaks the refinery down into its respective areas with a mass balance over each section.

The balance indicates that the largest section of the copper will be locked up in the copper electrowinning tank house as well as the low grade and high grade PLS and

electrolyte ponds. To reduce the locked-up copper in the circuit, the ponds can be run at lower levels, but that will decrease the buffer for the SX EW circuit as soon as there are upstream problems in the leach plant.

The results indicate that the blending cannot purely be based on the geological classification of the minerals. But a chemical, and preferably a mineralogical analysis of the stockpiles would be required to accurately predict the recovery of the mineral if is it to be leached solely under low pH conditions, without careful control of the Eh and Fe^{2+}/Fe^{3+} ratios. Further reductive leaching test work would be required to accurately model the and predict the recoveries under low pH reductive leaching conditions.

6 Preliminary cost analysis

Although not much data was available for a detailed financial feasibility study, the following method to determine the cut-off grade for the mine was disclosed. The calculation for the cut-off grade is a function of the various costs incurred during mining and the recovery of the concentrator and hydrometallurgical refinery. Equation 14, Equation 15 and Equation 16 explains the process followed to arrive at the cut-off grade for the operation. The cut-off grade was provided as 0.65% total copper contained in the ore.

$$Revenue = (Cu\ Price - Selling\ Cost - Processing\ Cost - Admin - Royalties) \times Recovery \times \frac{Ore}{Cu} Factor$$

Equation 14

$$Cost = Mining\ Cost + Processing\ Cost$$

Equation 15

$$Cut\ Off\ Grade = \frac{Cost}{Revenue}$$

Equation 16

Due to the high-grade ore that is available in the region, the operation always has a high profit margin if it is calculated based on the mining, refining and reagents cost. But other costs such as royalties, salaries, taxes etc. which are not available, thus an accurate financial study is not possible in this thesis.

However, the cost of acid accounts for approximately 50% of the operational budget of the refinery. The profit margin can be significantly improved if the acid consumption can be controlled. Based on the findings in the study about the calcium and acid consumption correlation and using the delivered to site price of 98% concentrated sulfuric acid of $350/t, Figure 32 could be sketched. The same assumptions was used as described in the Characterisation of leach performance of different lithologies section.

Figure 32 plots the increasing cost of acid per day as the contained calcium in the ore increases. The acid consumed by the gangue minerals is the amount of acid that needs to be trucked to the refinery. From the linear equation found in the Characterisation of leach performance of different lithologies section in Figure 27 Equation 12 was derived and used to determine the financial implication if the

contained calcium content would not be monitored. The budget value was a provided parameter. This figure was based on the available supply of acid and the desired feed rate of ore into the leach plant.

The figure clearly illustrates the financial impact that high calcium containing feed material could have on the operation.

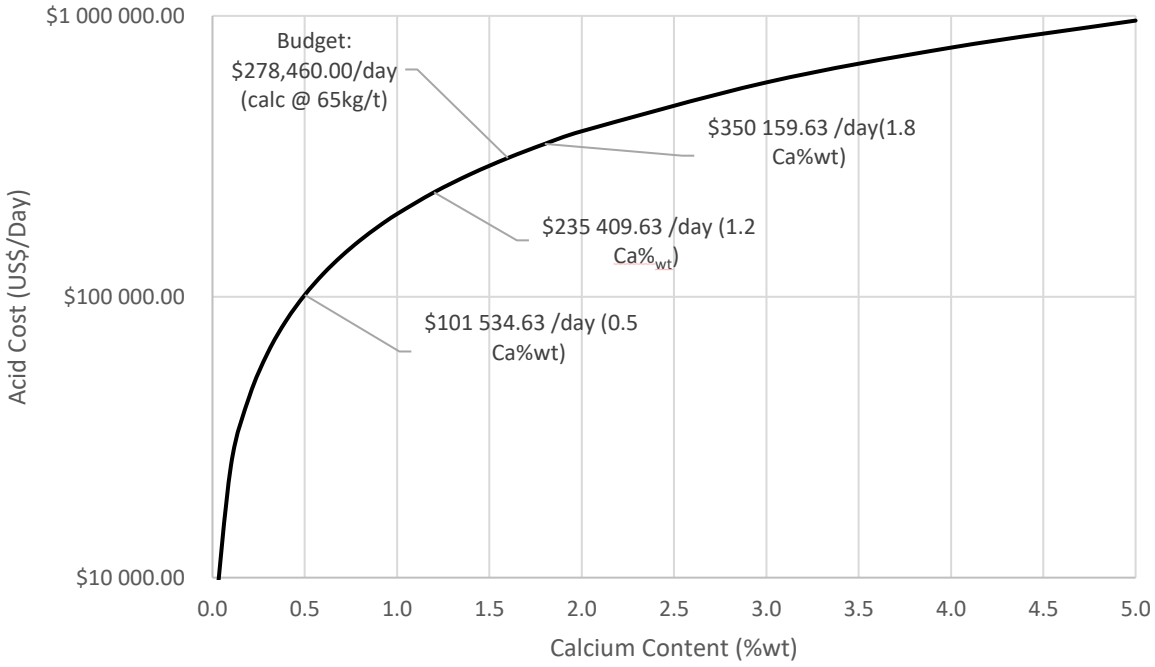


Figure 32: Cost of acid as a function of contained calcium in the ore

7 Conclusions and recommendations

7.1 Conclusion

The particle size test indicates that when the particle size is increased from P_{80} of 75µm to 150µm that there is no significant decrease in the recovery of sulphide material during the flotation process, nor is there a significant reduction in the leach recovery of the copper oxides. However, a significant decrease in the acid consumption was observed. When the particle size was further increased to a P_{80} of 212µm the average pre-flotation recovery dropped by 20.2%. The increased particle size reduced the copper oxide recovery by 4.1%, with little change to the total acid consumption.

Increasing the particle size reduced the flotation performance, as well as the copper oxide recovery. But the acid consumption values were significant enough to justify a 150µm blend instead of a 75µm blend. The blending of the individual lithologies could be seen as a key factor in the performance of the hydrometallurgical refinery.

Test results obtained from blended ore samples which were based on the mine plan, indicated that acid concentration, measured as pH, had the greatest effect on the leach recovery and particularly the acid consumption. The FAC decreased from 142kg/MT to 86kg/MT for leach tests performed at pH values of 1.0 and 2.0 respectively. The copper recovery decreased very slightly from 95.4%$_{CuOx}$ to 93%$_{CuOx}$ for leach tests performed at a pH value of 1.0 and 2.0 respectively. The effect on cobalt however was much more pronounced as the total cobalt recovery dropped from 82% to 60% for pH values of 1.0 and 2.0 respectively.

A change in the percentage solids in the leach slurry showed that the optimal leach conditions was at 30% solids. As the leach recovery dropped when from 99.6% to 97.6% and 97.0% for 20% and 40% solids respectively. The temperatures of the leach tests were analysed at 30°C, 45°C and 60°C with very little effect on the copper oxide leach efficiency that changed from 98.2%, 98.2% and 98.5% respectively. The small effect caused by the temperature indicated that the leach reactions were not chemical reaction limiting. This was confirmed by the leach kinetics tests where the product diffusion limiting model suited the results best.

As seen with the pH tests, the cobalt was found to be much more affected by the change in temperature and varied from 78.6% to 88.0% for tests conducted at 30°C and 60°C respectively. The change in cobalt recovery could be due to observed pH in the system caused by the change in temperature.

From 208 leach tests conducted on various lithologies it was determined that the measurement of the contained calcium is a very good indicator of the gangue acid consumption. That can be expected from any ore sample and follows a linear directly proportional trend, approximating an increase of 45kg of acid per metric ton of ore treated per percent increase in calcium contained in the ore sample. The following equation describes the GAC as a function of contained calcium in the feed.

$$GAC\left(\frac{kg_{H_2SO_4}}{DMT_{ore}}\right) = 44.667 \times Ca\%_{wt} - 1.0863$$ *Equation 17*

To determine the copper oxide recovery, a correlation was found between the contained cobalt in the feed sample and the copper oxide recovery. There was an indirect proportional correspondence between the increase in cobalt in the ore and the copper oxides recovered during the leach process. A linear equation fitted the data best, but only for values greater than 0.35%$_{TCo}$. The following correlation was found for the copper vs leach recovery curve.

$$Cu\ Oxide\ Rec|_{3.00\%_{TCo}}^{0.35\%_{TCo}} = -0.0151(Total\ Co\ Contained) + 0.9863$$ *Equation 18*

If a model was to be developed to predict the performance of the expected leach recovery and gangue acid consumption of the material mined, a combination of Equation 17 and Equation 18 could be used.

7.2 Recommendations

1. A more detailed study on the effect of contaminants in raffinate, such as Ca, Mg, Fe and Al could have on the recovery and leach kinetics of copper and cobalt.

2. Due to the sensitivity of the GAC to calcium variations, online XRD analysers could be installed on reclaim conveyers from the stockpiles to calculate the expected GAC values expected from the blend. This would allow the blend to be altered before very high acid consuming ore reached the hydro refinery.

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Appendix A: Nomenclature and glossary

Symbol	Description
AAS	Atomic Absorption Spectrometer
Co	Cobalt
Cu	Copper
CuS	Copper Sulphides
DMT	Dry Metric Ton
DSTRAT	Dolomitic stratification
EW	Electrowinning
FAC	Fresh Acid Consumption
FAM	Iron, Aluminium & Manganes
Fe	Iron
FTAL	Flotation Tailing Acid Leach
GAC	Gangue Acid Consumption
HG	High Grade
ICP-OES	Inductively Coupled Plasma – Optical Emission Spectrometer
KOV	Kamoto Oliveira Virgule
LG	Low Grade
ME	Mashamba East
MT	Metric Ton
NaSH	Sodium hydrogen sulphide
OBI	Ore Body Inferior
OBS	Ore Body Superior
P&ID	Process and Instrumentation Diagram
PFD	Process Flow Diagram
PLS	Pregnant Leach Solution
Ratgris	Roches Argilo-Talqueuses Grey
Ratlillas	Roches Argilo-Talqueuses lilas
RSC	Roches Siliceuses Cellulaires
RSF	Roches Siliceuses Feuilletees
SCM	Shrinking Core Model
SDB	Schistes Dolomitiques

Symbol	Description
SDS	Schistes Dolomitiques Superior
SX	Solvent Extraction
TAC	Total Acid Consumption

Appendix B: Refinery Copper Balance

Unit Description	Solids in Tank (DMT)	Aqueous in Tank (MT)	Copper in Solids ($\%_{wt}$)	Copper in Aqueous ($\%_{wt}$)	Copper Contained (DMT)
23 - Sulphide concentrate thickening and pressure filtration (2-15)					
Sulphide concentrate thickener no 1 (X4)	**349**	**1156**	**37%**	**0%**	**129**
Sulphide concentrate thickener no 2 (Y4)	**349**	1156	37%	0%	129
KTC return water tank (existing Y5 at sulphide thickeners)	**0**	1478	37%	0%	0
Sulphide concentrate surge tank no 2	**985**	519	37%	0%	365
Section Total	1684	**4309**			**629**
24 - Oxide ore receiving and preleach thickening					
Oxide or receiving surge tank	**492**	896	4%	0%	18
Oxide ore thickener feed distribution tank	**492**	896	4%	0%	18
Oxide ore preleach thickener no 1	**3478**	2319	4%	0%	127
Oxide ore preleach thickener no 2	**3478**	2319	4%	0%	127
Preleach thickener common needle tank	**0**	329	4%	0%	0
Section Total	7940	**6758**			**290**
25 - Oxide concentrate wash displacement (THERE ARE TWO IDENTICAL TRAINS !)					
No 1 displacement wash thickener preleach tank	**484**	995	3%	1%	48
No1 displacement wash thickener	**1986**	4244	3%	1%	198
No 2 displacement wash thickener mixing tank	**14**	30	4%	1%	2
No 2 displacement wash thickener	**1918**	4220	4%	1%	215
No 3 displacement wash thickener mixing tank	**0**	36	4%	1%	1
No 4 displacement wash thickener mixing tank	**13**	29	4%	0%	1
No 4-displacement wash thickener	**1779**	4167	4%	0%	184
Displacement wash needle tank	**0**	35	4%	0%	0
Section Total	6194	**13756**			**650**
28 - No 3-sulphide roaster (OLD HATCH) (31-42)					
No 1 Roaster slurry feed tank	**15**	8	37%	0%	6
No 1 Roaster quench tank	**6**	84	29%	1%	3
Section Total	22	**92**			**8**

Unit Description	Solids in Tank (DMT)	Aqueous in Tank (MT)	Copper in Solids ($\%_{wt}$)	Copper in Aqueous ($\%_{wt}$)	Copper Contained (DMT)
29 - No 4-sulphide roaster (NEW HATCH) (44-55)					
No 2 Roaster slurry feed tank	**15**	8	37%	0%	6
No 2 Roaster quench tank	**6**	84	29%	1%	3
Section Total	22	**92**			**8**
34 - Raffinate reticulation and calcine surge (93-102)					
No 1 raffinate storage tank (4R5)	**0**	272	0%	0%	1
No 2 Raffinate circulation tank (4R6)	**0**	272	0%	0%	1
Calcine leach feed surge tank	**16**	212	29%	1%	7
Section Total	16	**756**			**8**
35 - New leach (ONLY ONE OF TWO LEACH TRAINS SHOWN !!)					
Leach acid distribution tank	**0**	664	0%	0%	2
Leach feed distribution tank	**882**	1078	4%	1%	40
No 1 leach tank train 1	**643**	958	2%	2%	61
No 2 leach tank train 1	**551**	1011	2%	2%	54
No 3 leach tank train 1	**500**	1041	1%	2%	50
No 4 leach tank train 2	**467**	1060	1%	2%	47
No 5 leach tank train 2	**446**	1072	1%	2%	46
No 6 leach tank train 3	**430**	1081	0%	2%	45
Section Total	3920	**7966**			**344**
36 - New leach primary thickening and clarification (ONY ONE OF TWO TRAINS ARE SHOWN !!)					
New whole ore leach primary thickener	**1720**	4635	0%	2%	188
HG PLS Coagulant flash mix tank	**0**	39	0%	2%	1
New whole ore leach HG clarifier feed tank	**0**	260	0%	2%	10
HG clarifiers	**1**	4565	0%	2%	170
New whole ore LG clarifier feed tank	**0**	198	0%	1%	5
LG clarifiers	**1**	4334	0%	1%	104
Section Total	1721	**14030**			**478**
37 - A SX Ponds					
1st compartment dirty HG PLS pond	**0**	15903	0%	2%	297
2nd compartment dirty HG PLS pond	**0**	15918	0%	2%	298
1st compartment clean HG PLS pond	**0**	15932	0%	2%	299
2nd compartment clean HG PLS pond	**0**	15948	0%	2%	300
Pin bed clarifier feed tank (old o/f tank)	**0**	131	0%	2%	2
HG pin bed clarifier no 1	**0**	545	0%	0%	0

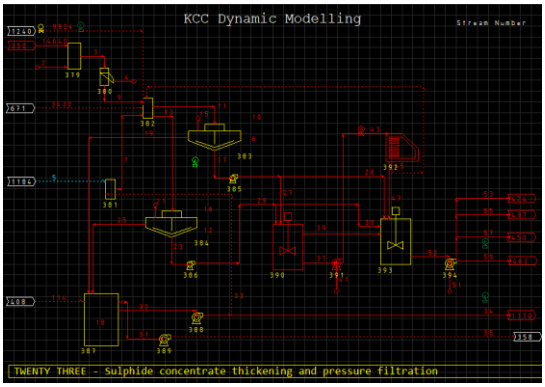
Unit Description	Solids in Tank (DMT)	Aqueous in Tank (MT)	Copper in Solids ($\%_{wt}$)	Copper in Aqueous ($\%_{wt}$)	Copper Contained (DMT)
HG pin bed clarifier no 2	**0**	545	0%	0%	0
Pin bed clarifier u/f tank	**2**	3	0%	2%	0
1st compartment dirty LG PLS pond	**0**	15097	0%	1%	181
2nd compartment dirty LG PLS pond	**0**	15106	0%	1%	182
1st compartment clean LG PLS pond	**0**	15119	0%	1%	182
2nd compartment clean HG PLS pond	**0**	15123	0%	1%	183
Section Total	4	**125369**			**1923**
39 - Counter current decantation (CCD)(THERE ARE TWO IDENTICAL TRAINS !) (NEEDLE TANKS AND PUMPS NOT SHOWN)					
No 1a CCD thickener mixing tank	**20**	26	0%	0%	0
No 1a CCD thickener existing 6C5	**2667**	3635	0%	0%	28
No2a CCD thickener wash tank	**19**	26	0%	0%	0
No 2a CCD wash thickener existing 6C4	**2588**	3686	0%	0%	32
No 3a CCD thickener wash tank	**18**	26	0%	0%	0
No 3a CCD wash thickener existing 6C3	**2517**	3737	0%	0%	37
CCD 4a thickener mixing tank	**18**	27	0%	0%	0
CCD 4z wash thickener existing 6C2	**2454**	3792	0%	0%	45
CCD 5a mixing tank	**18**	27	0%	0%	0
CCD 5a wash thickener existing 6C1	**2310**	3899	0%	0%	57
CCD 6a thickener mixing tank	**16**	28	0%	1%	1
CCD 6z wash thickener	**2247**	4003	0%	1%	77
CCD 7a thickener mixing tank	**15**	30	0%	1%	1
CCD 7z wash thickener	**2067**	4234	0%	1%	119
Section Total	16974	**27176**			**398**
41 - High grade solvent extraction (187-222)					
Train one loaded organic circulation tank	**0**	370	0%	2%	17
SX train one extraction stage 2 mixer settler	**0**	152	0%	2%	7
SX train one extraction stage 2 mixer settler	**0**	304	0%	2%	10
SX train one extraction stage 2 mixer settler	**0**	152	0%	2%	6
SX train one extraction stage 2 mixer settler	**0**	301	0%	1%	6
SX train one extraction stage 2 mixer settler	**0**	152	0%	1%	4
SX train one extraction stage 2 mixer settler	**0**	298	0%	0%	2
SX train one strip stage 1 mixer settler	**0**	152	0%	1%	2

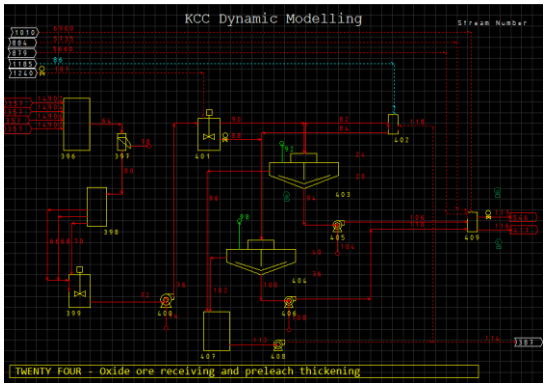
Unit Description	Solids in Tank (DMT)	Aqueous in Tank (MT)	Copper in Solids ($\%_{wt}$)	Copper in Aqueous ($\%_{wt}$)	Copper Contained (DMT)
SX train one strip stage 1 mixer settler	**0**	308	0%	3%	20
SX train one strip stage 2 mixer settler	**0**	152	0%	1%	3
SX train one strip stage 2 mixer settler	**0**	314	0%	4%	27
SX Raffinate pond	**0**	32078	0%	0%	83
Section Total	0	**34736**			**186**
42 - Low grade solvent extraction (187-222)					
SX train two extraction stage one mixer settler	**0**	138	0%	2%	3
SX train two extraction stage one mixer settler	**0**	262	0%	1%	2
SX train two extraction stage 2 mixer settler	**0**	138	0%	2%	3
SX train two extraction stage 2 mixer settler	**0**	260	0%	0%	1
SX train two extraction stage 3 mixer settler	**0**	138	0%	1%	2
SX train two extraction stage 3 mixer settler	**0**	259	0%	0%	0
SX train two strip stage 1 mixer settler	**0**	162	0%	1%	1
SX train two strip stage 1 mixer settler	**0**	329	0%	3%	10
SX train two strip stage 2 mixer settler	**0**	162	0%	1%	1
SX train two strip stage 2 mixer settler	**0**	335	0%	4%	14
SX Raffinate pond	**1**	30727	0%	0%	83
Section Total	1	**32910**			**122**
43 - Dual media filters for electrolyte					
Anolyte return tank	**0**	1264	0%	4%	55
Catholyte return tank	**0**	1270	0%	4%	55
Anolyte strip liquor return tank	**0**	1251	0%	3%	37
Section Total	0	**3785**			**146**
44 - Primary copper electrowinning EW2 (New tankhouse) (224-256)					
No 1 Advanced electrolyte tank	**0**	405	0%	4%	17
NO 2 Advanced electrolyte tank	**0**	207	0%	4%	9
Copper Scavenger cells (new tankhouse)	**0**	1202	0%	4%	49
Primary copper electrowinning cells (South)	**0**	1177	0%	3%	35
Primary copper electrowinning cells (South)	**0**	1177	0%	3%	35
Primary copper electrowinning cells (Central)	**0**	1177	0%	3%	35

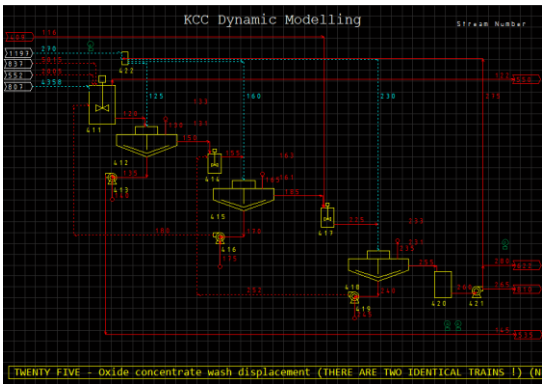
Unit Description	Solids in Tank (DMT)	Aqueous in Tank (MT)	Copper in Solids ($\%_{wt}$)	Copper in Aqueous ($\%_{wt}$)	Copper Contained (DMT)
Primary copper electrowinning cells (Central)	**0**	1177	0%	3%	35
Commercial EW recirculation tank 1	**0**	111	0%	4%	4
Commercial EW recirculation tank 4	**0**	393	0%	3%	13
Commercial EW recirculation tank 7	**0**	393	0%	3%	13
Commercial EW recirculation tank 2	**0**	201	0%	3%	6
Commercial EW recirculation tank 5	**0**	766	0%	3%	23
Commercial EW recirculation tank 8	**0**	212	0%	3%	6
Commercial EW recirculation tank 3	**0**	111	0%	4%	4
Commercial EW recirculation tank 6	**0**	393	0%	3%	13
Commercial EW recirculation tank 9	**0**	393	0%	3%	13
Copper on Cathodes	**0**	9495		2160000 %	1080
Section Total					**1389**
45 - Primary copper electrowinning EW3 (New tankhouse) Senet (258-285)					
TH1 electrolyte circulation tank rich side	**0**	161	0%	3%	5
TH1 electrolyte circulation tank lean side	**0**	160	0%	3%	5
TH2 electrolyte circulation tank rich side	**0**	160	0%	3%	5
TH2 electrolyte circulation tank lean side	**0**	159	0%	3%	5
Copper Scavenger cells (TH1 Senet)	**0**	251	0%	4%	10
Primary copper electrowinning cells (TH1 Senet)	**0**	246	0%	3%	7
Primary copper electrowinning cells (TH1 Senet)	**0**	246	0%	3%	7
Copper Scavenger cells (TH2 Senet)	**0**	251	0%	3%	7
Primary copper electrowinning cells (TH2 Senet)	**0**	246	0%	3%	7
Primary copper electrowinning cells (TH2 Senet)	**0**	246	0%	3%	7
Copper on Cathodes	**0**	2124		3456000 %	1728
Section Total					**1795**
47 - Cobalt bleed stream iron and copper removal precipitation tanks					
No 1 iron removal tank	**31**	581	1%	0%	3
No 2 iron removal tank	**35**	578	1%	0%	3
No 3 iron removal tank	**37**	575	1%	0%	3
No 1 iron removal tank	**31**	581	1%	0%	3
No 2 iron removal tank	**35**	578	1%	0%	3
No 3 iron removal tank	**37**	575	1%	0%	3

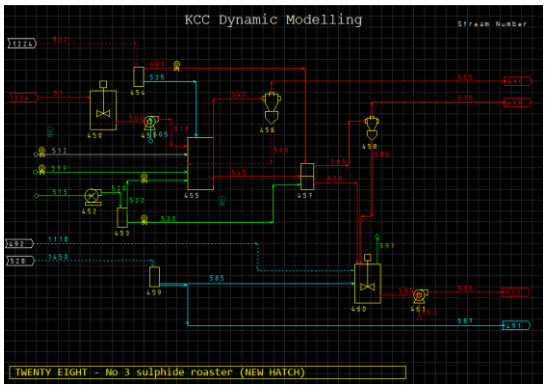
Unit Description	Solids in Tank (DMT)	Aqueous in Tank (MT)	Copper in Solids ($\%_{wt}$)	Copper in Aqueous ($\%_{wt}$)	Copper Contained (DMT)
No 1 cobalt bleed stream primary copper removal tank	**38**	569	15%	0%	7
No 2 cobalt bleed stream primary copper removal tank	**41**	566	15%	0%	7
No 3 cobalt bleed stream primary copper removal tank	**43**	564	15%	0%	6
Section Total	327	**5168**			**37**
48 - Iron Aluminium and copper removal thickening and filtration					
Iron removal thickener (Z4) feed tank	**5**	72	1%	0%	0
Iron removal thickener (Z4)	**275**	4460	1%	0%	23
Iron / Aluminium removal thickener overflow transfer tank	**0**	323	1%	0%	1
Iron Aluminium removal thickener underflow tank	**131**	197	1%	0%	2
Copper removal thickener (Z6) feed tank	**6**	74	15%	0%	1
Copper removal thickener (Z6)	**196**	2699	15%	0%	30
Copper removal thickener overflow transfer tank	**0**	610	15%	0%	0
Copper removal filter feed tank	**160**	374	15%	0%	24
Section Total	772	**8808**			**81**
49 - Primary Cobalt precipitation with Magnesia for salt Secondary cobalt precipitation with lime					
No 1 Primary Cobalt precipitation tank	**21**	1350	3%	0%	1
No 2 Primary cobalt precipitation tank	**23**	1349	3%	0%	1
No 3 Primary cobalt precipitation tank	**24**	1348	3%	0%	1
No 4 Primary cobalt precipitation tank	**25**	1347	3%	0%	1
Section Total	93	**5394**			**3**

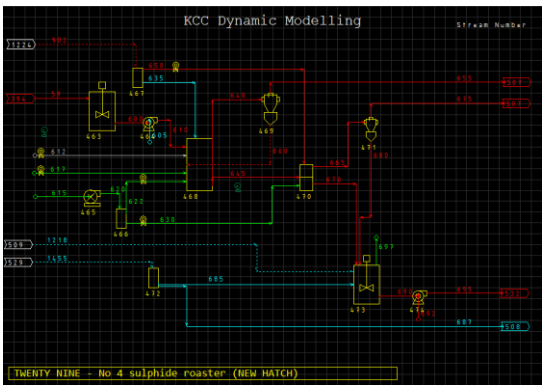
Appendix C: Refinery Copper Balance Flowsheet

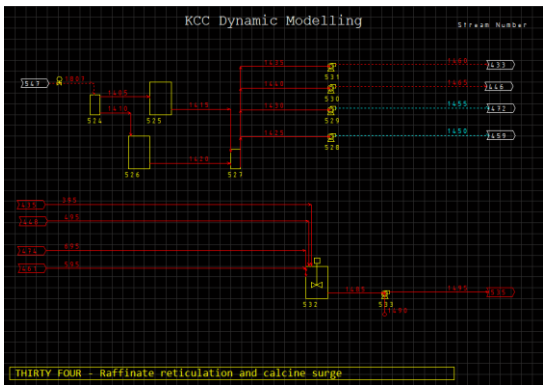


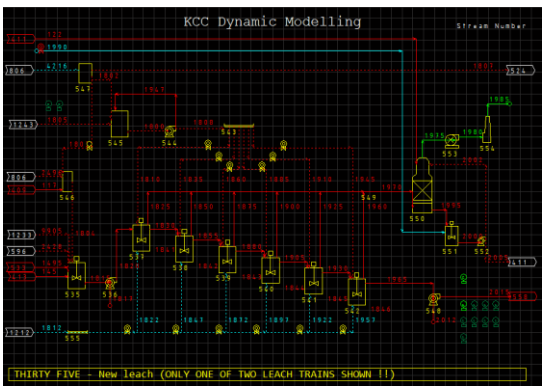


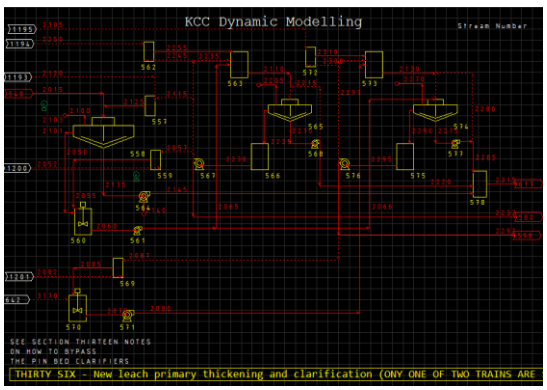


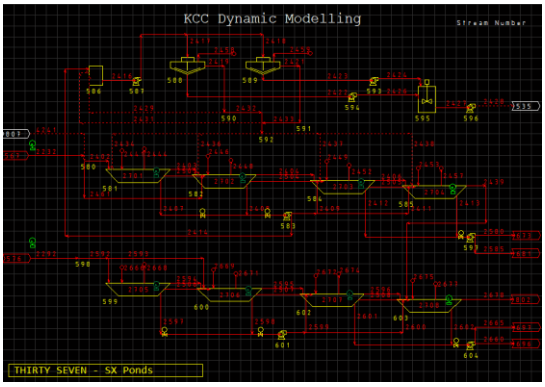


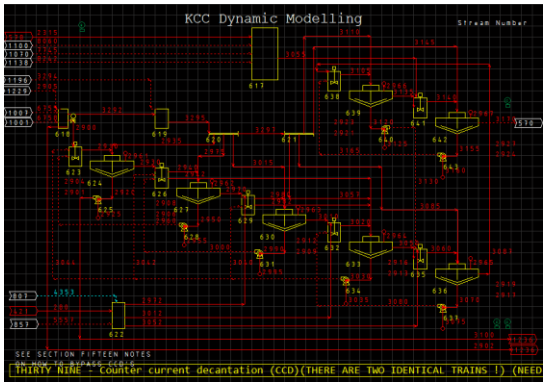


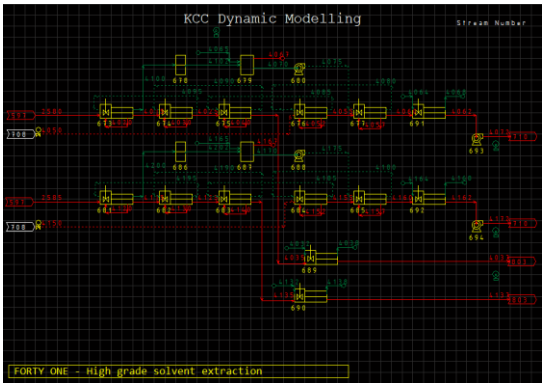


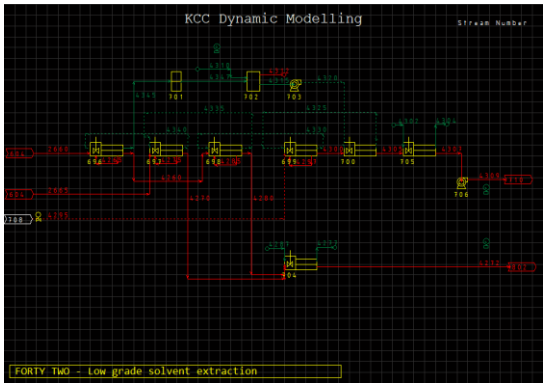


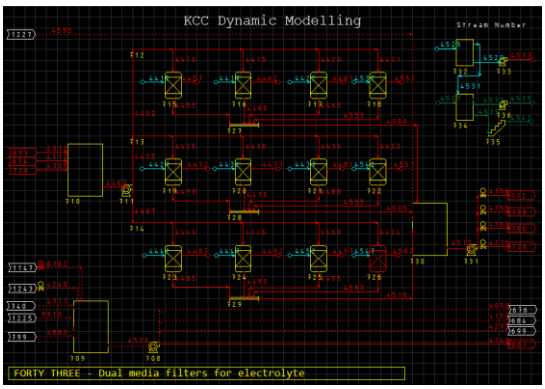


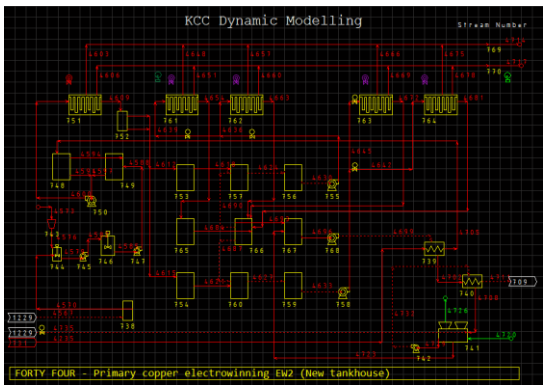


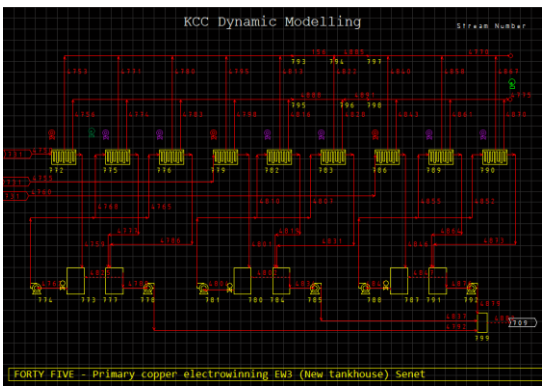


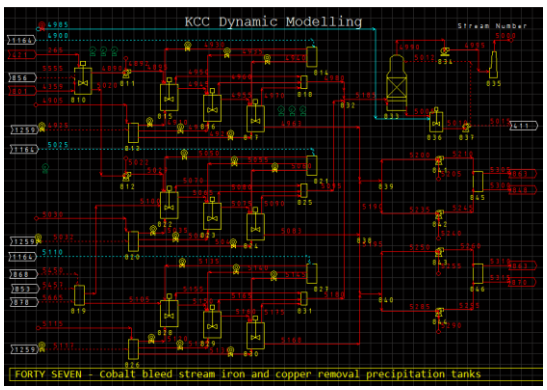


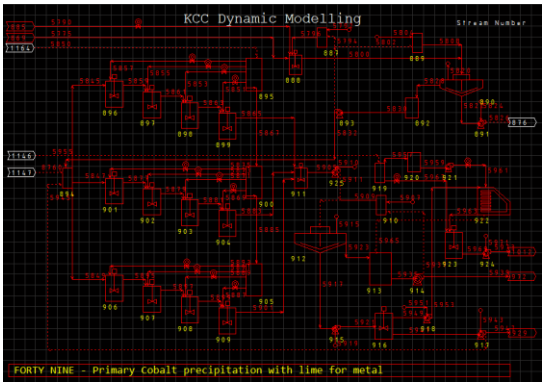


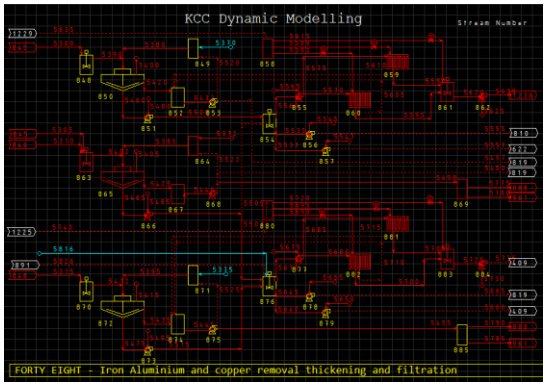












Appendix D: Test Results

Initial particle size tests

Table 24: FNSR 150µm leach test results

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Test ID	**WOL-T-10**		**WOL-T-08**		**WOL-TO6**		**WOL-T09**		**WOL-T-04**		**WOL-T-03**		**WOL-T-05**	
Samples	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**
Time (minutes)	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**
0	1.459	1.438	1.480	1.471	1.435	1.401	1.397	1.46	1.389	1.348	1.520	1.51	1.410	1.50
10	1.533	1.650	1.522	1.483	2.252	1.972	1.473	1.65	2.126	2.046	1.535	1.55	1.557	1.56
20	1.422	1.211	1.490	1.549	1.569	2.179	1.513	1.49	1.666	1.501	1.551	1.51	1.505	1.51
30	1.497	1.479	1.503	1.500	1.613	1.533	1.501	1.51	1.808	1.515	1.532	1.52	1.479	1.52
60	1.490	1.500	1.473	1.507	1.434	1.521	1.475	1.51	1.494	1.501	1.495	1.53	1.442	1.52
90	1.490	1.504	1.566	1.492	1.508	1.432	1.493	1.49	1.492	1.500	1.507	1.49	1.513	1.51
120	1.525	1.474	1.516	1.475	1.520	1.363	1.474	1.41	1.503	1.505	1.497	1.50	1.518	1.51
150	1.481	1.502	1.504	1.496	1.496	1.512	1.487	1.55	1.488	1.490	1.511	1.49	1.491	1.51
180	1.504	1.481	1.499	1.501	1.463	1.508	1.488	1.48	1.494	1.507	1.486	1.50	1.416	1.50
210	1.466	1.478	1.503	1.506	1.485	1.504	1.506	1.51	1.492	1.476	1.494	1.50	1.471	1.49
240	1.492	1.475	1.506	1.511	1.499	1.494	1.490	1.51	1.465	1.486	1.502	1.50	1.507	1.50
Average	**1.5**	**1.5**	**1.5**	**1.5**	**1.6**	**1.6**	**1.5**	**1.5**	**1.6**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**
Time (minutes)	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**
0	322.9	329.8	332.3	321.3	327.9	332.8	323.3	299.6	290	296.1			321.6	299.8
10	318.2	317.5	319.6	331.2	279.8	299.2	322.1	290.5	322.1	334.7	305.9	290.8	312.6	297.9
20	324.0	342.0	321.6	325.3	320.3	284.9	320.1	299.0	338.4	338.5	307.2	296.6	314.4	298.2
30	318.7	324.0	320.0	327.1	317.2	323.9	319.3	296.7	328.9	330.3	307.8	293.8	315.5	296.6
60	320.1	324.0	319.5	323.3	325.1	322.4	318.7	294.2	322.5	324.0	309.3	292.2	314.4	294.6
90	317.7	331.7	322.3	329.2	318.1	329.8	315.6	293.6	322.0	324.1	308.4	295.1	308.4	294.9
120	319.0	326.5	334.8	339.5	322.2	333.0	319.6	303.1	318.3	326.4	309.8	295.7	310.7	296.2
150	318.5	324.7	326.3	329.0	323.2	323.2	316.8	292.2	323.5	323.9	310.1	296.4	315.2	293.4
180	319.0	326.4	321.3	323.6	321.4	331.7	316.5	296.6	324.8	320.3	311.6	296.0	316.1	295.4
210	321.0	326.5	319.1	325.3	327.4	324.8	317.7	296.6	318.0	321.5	310.3	295.8	310.6	296.5
240	316.8	324.9	316.4	321.8	324.4	322.6	316.3	293.8	322.0	3210 6.0	308.3	294.4	309.0	293.9
Average	**320**	**327**	**323**	**327**	**319**	**321**	**319**	**296**	**321**	**3213**	**309**	**295**	**314**	**296**
H2SO4 conc (g/l)	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	501.76	501.76	501.76	501.76	602.82	602.82

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Sample weight (g)	400.18	400.28	400.00	400.00	400.00	400.00	400.07	400.09	400.00	400.00	400.00	400.00	400.00	400.00
Acid to pH @ t=0 (ml)	5.30	6.90	5.10	6.20	5.80	7.60	5.10	4.50	3.10	3.40	3.00	3.60	4.10	3.90
Total Leach acid (ml)	60.80	58.50	172.80	169.90	135.20	131.40	72.20	72.10	105.50	105.00	77.40	79.90	68.60	68.30
PLS volume (ml)	990.00	980.00	970.00	995.00	885.00	830.00	930.00	890.00	968.00	937.00	945.00	920.00	915.00	930.00
PLS weight (g)	1018.00	1003.10	1026.50	1058.00	969.30	901.60	960.50	922.80	1045.00	1113.00	992.20	969.10	941.30	964.10
Wash water volume (ml)	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1100.00	1100.00	1000.00	1000.00	1000.00	1000.00
Wash water weight (g)														
Wash filtrate volume (ml)	985.00	980.00	1085.00	990.00	975.00	1060.00	1010.00	970.00	1000.00	930.00	981.00	955.00	980.00	980.00
Wash filtrate weight (g)	988.00	996.40	1088.00	995.10	991.80	1072.50	1014.50	973.90	1003.40	935.00	981.30	957.00	981.30	984.80
Filter paper weight (g)	3.57	3.61	3.56	3.57	3.61	3.59	3.57	3.56	3.54	3.54	3.60	3.58	3.61	3.62
Wet cake + filter paper (g)	490.40	528.90	533.60	534.40	550.10	550.04	534.30	568.70	451.60	443.60	489.40	463.20	517.30	504.70
Dry cake weight (g)	383.70	384.31	358.61	361.03	348.89	343.98	376.93	376.79	337.60	333.17	359.82	358.27	378.08	374.66
Feed solids (S-samples)														
T-Cu, %	0.5575	0.5657	1.9879	2.01	6.2472	6.2013	2.1101	2.1784	8.3282	8.2354	4.5481	4.4938	1.6336	1.7297
OX-Cu, %	0.55	0.56	1.8082	1.8677	5.01	4.9	2.03	2.1	7.75	7.65	4.45	4.4	1.6	1.65
% OX-Cu	99%	99%	91%	93%	80%	79%	96%	96%	93%	93%	98%	98%	98%	95%
T-Co, %	0.0797	0.0908	0.2305	0.15	0.0984	0.0906	0.4337	0.4626	0.3487	0.3406	0.2342	0.2392	0.2301	0.2086
OX-Co, %	0.07	0.09	0.1459	0.1236	0.08	0.09	0.43	0.46	0.33	0.33	0.18	0.16	0.2074	0.18
Fe, %	4.2063	5.1192	0.7988	0.801	0.5756	0.5617	1.9808	2.0724	0.7542	0.7284	0.4532	0.4517	0.2147	0.2049
Leach residue (C-samples)														
T-Cu, %	0.0993	0.0869	0.2829	0.25	1.2963	1.3493	0.268	0.2156	0.4837	0.4642	0.24	0.1487	0.0748	0.0942
OX-Cu, %	0.09	0.08	0.1282	0.1522	0.34	0.35	0.26	0.19	0.25	0.22	0.2	0.14	0.0646	0.08
% OX-Cu	91%	92%	45%	61%	26%	26%	97%	88%	52%	47%	83%	94%	86%	85%
T-Co, %	0.0667	0.06487	0.1601	0.09	0.0656	0.0737	0.3687	0.382	0.1596	0.1678	0.1579	0.1502	0.2342	0.1554
OX-Co, %	0.06	0.05	0.0821	0.057	0.04	0.04	0.37	0.39	0.13	0.13	0.08	0.09	0.1655	0.12
Fe, %	4.4394	5.0071	0.8495	0.853	0.5182	0.5396	2.0012	2.1321	0.8631	0.8835	0.4627	0.4608	0.1841	0.1876
PLS (PLS-samples)														
pH	1.68	1.55	1.42	1.57	1.72	1.05	1.72	1.77	1.5	1.46	1.72	1.56	1.39	1.55
Free Acid g/L	3.76	4.94	4.55	4.55	4.091	10.43	3.72	3.76	4.16	4.16	2.59	3.71	3.76	3.63
Cu (g/L)	1.76	1.72	5.7562	5.95	17.37	17.23	6.92	7.2	29.37	29.26	17.43	17.48	6.7367	6.7
Co (g/L)	0.24	0.22	0.3858	0.43	0.38	0.32	0.48	0.43	1.04	0.94	0.45	0.51	0.359	0.5
Fe (g/L)	0.12	0.13	0.26	0.23	0.36	0.39	0.16	0.14	0.15	0.14	0.22	0.18	0.29	0.32
Wash water (WW-samples)														

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
pH	2.03	1.75	1.91	2.07	2.31	1.86	2.23	2.02	1.82	1.83	2.07	1.9	2.66	2.15
Free Acid g/L	0.01	1.41	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Cu (g/L)	0.21	0.26	0.9769	0.95	2.83	4.37	0.88	1.33	3.2	4.05	2.14	1.92	0.9048	0.66
Co (g/L)	0.02	0.02	0.083	0.14	0.2	0.24	0.11	0.24	0.16	0.2	0.1	0.13	0.1026	0.18
Fe (g/L)	0.01	0.02	0.0431	0.1	0.1	0.15	0.04	0.07	0.05	0.17	0.06	0.17	0.0314	0.07
Cu Accountability	**104 %**	**100 %**	**96%**	**97%**	**91%**	**95%**	**99%**	**98%**	**100 %**	**99%**	**107 %**	**103 %**	**112 %**	**105 %**
Leach Efficiencies														
T-Cu recovery (solids)	83%	85%	87%	89%	82%	81%	88%	91%	95%	95%	95%	97%	96%	95%
OX-Cu recovery (solids)	84%	86%	94%	93%	94%	94%	88%	91%	97%	98%	96%	97%	96%	95%
T-Co recovery (solids)	20%	31%	38%	46%	42%	30%	20%	22%	61%	59%	39%	44%	4%	30%
OX-Co recovery (solids)	18%	47%	50%	58%	56%	62%	19%	20%	67%	67%	60%	50%	25%	38%
Fe recovery (solids)	-1%	6%	5%	4%	21%	17%	5%	3%	3%	-1%	8%	9%	19%	14%
Solids mass reduction, %	4.1%	4.0%	10.3 %	9.7%	12.8 %	14.0 %	5.8%	5.8%	15.6 %	16.7 %	10.0 %	10.4 %	5.5%	6.3%
TAC (kg/t - ore))	90.24	82.88	257.05	254.05	203.42	187.81	107.80	107.02	126.13	126.20	94.71	96.18	100.94	100.34
GAC (kg/t-ore) [Cu Solids]	83.11	75.43	230.28	226.51	124.45	110.01	79.13	76.54	3.89	5.06	27.85	28.88	76.81	75.01
FAC (kg/t-ore) [Cu Solids]	85.54	77.96	239.38	235.87	151.30	136.46	88.88	86.90	45.46	46.25	50.58	51.76	85.02	83.62

Table 25: Table 18: FNSR 212µm leach test results

P_{80}	212													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Test ID	**WOL - T - 20**		**WOL - T - 18**		**WOL - T - 17**		**WOL - T - 19**		**WOL - T - 16**		**WOL - T - 14**		**WOL - T - 15**	
Samples	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**
Time (minutes)	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**
0	1.520	1.440	1.490	1.510	1.460	1.438	1.482	1.385	1.510	1.470	1.420	1.460	1.506	1.483
10	1.670	1.580	1.500	1.500	1.540	1.698	1.480	1.482	1.800	1.940	1.530	1.450	1.492	1.612
20	1.500	1.520	1.600	1.590	1.714	1.655	1.542	1.543	1.480	1.380	1.490	1.500	1.530	1.535
30	1.500	1.520	1.510	1.520	1.766	1.447	1.482	1.374	1.470	1.500	1.380	1.540	1.524	1.514
60	1.500	1.510	1.510	1.520	1.504	1.489	1.494	1.502	1.290	1.200	1.500	1.500	1.503	1.506
90	1.500	1.490	1.520	1.520	1.506	1.514	1.534	1.503	1.340	1.250	1.510	1.490	1.513	1.504
120	1.500	1.500	1.480	1.490	1.408	1.509	1.504	1.508	1.380	1.370	1.510	1.490	1.538	1.517

P_{80}	212													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
150	1.500	1.490	1.480	1.490	1.428	1.501	1.492	1.497	1.430	1.400	1.500	1.490	1.501	1.491
180	1.500	1.500	1.490	1.510	1.496	1.499	1.482	1.485	1.440	1.400	1.480	1.490	1.488	1.499
210	1.490	1.480	1.490	1.490	1.506	1.506	1.487	1.462	1.480	1.390	1.490	1.480	1.501	1.500
240	1.500	1.500	1.500	1.500	1.495	1.487	1.503	1.499	1.490	1.410	1.500	1.500	1.500	1.504
Average	**1.516**	**1.503**	**1.506**	**1.513**	**1.529**	**1.522**	**1.498**	**1.476**	**1.465**	**1.428**	**1.483**	**1.490**	**1.509**	**1.515**
Time (minutes)	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**
0	326.5	325.4	320.1	315.5	317.4	320	315.8	322.4	321.9	321.1	328.3	323	313.4	317.8
10	324.4	326.4	331.4	329.1	323.4	317.8	327.7	330.2	317.7	305.4	330.3	330.7	321.4	315.7
20	335.3	331.0	328.0	330.0	312.5	322.4	321.9	327.0	338.7	341.2	333.2	329.0	321.2	323.0
30	335.2	331.2	333.7	335.7	310.3	336.6	325.5	337.4	339.2	334.4	339.6	327.0	321.4	326.1
60	334.1	330.3	333.0	334.5	326.7	333.4	323.2	327.7	346.0	348.5	330.5	327.9	321.2	324.1
90	331.4	329.0	331.5	331.2	325.7	330.7	322.3	324.9	340.2	342.9	328.5	329.6	321.7	323.4
120	329.5	326.6	331.9	330.7	329.6	330.1	330.9	327.8	337.0	343.0	330.5	328.2	319.7	324.6
150	330.3	331.2	330.6	328.8	326.7	326.7	326.5	326.4	336.4	340.2	330.6	326.5	322.0	327.2
180	329.2	330.5	342.0	334.9	321.3	326.0	324.7	325.6	333.9	337.3	332.0	325.6	323.2	325.8
210	330.8	329.1	343.9	333.3	326.0	324.3	322.4	325.5	334.1	334.7	331.4	330.9	323.4	326.9
240	335.8	328.6	338.9	330.3	323.2	326.1	323.3	327.5	331.6	333.7	329.0	329.7	321.7	324.6
Average	**331.1**	**329.0**	**333.2**	**330.4**	**322.1**	**326.7**	**324.0**	**327.5**	**334.2**	**334.8**	**331.3**	**328.0**	**320.9**	**323.6**
H2SO4 conc (g/l)	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79
Sample weight (g)	400.01	400.20	400.04	400.11	400.04	400.00	400.06	400.12	400.65	400.19	400.11	400.11	400.36	400.01
Acid to pH @ t=0 (ml)	4.80	4.90	4.60	4.10	4.20	5.30	3.30	5.40	4.30	4.50	6.00	4.20	4.20	5.40
Total Leach acid (ml)	66.30	62.90	147.10	137.90	197.20	201.00	106.10	103.90	100.00	100.00	74.00	74.00	66.00	67.50
PLS volume (ml)	945.00	960.00	870.00	915.00	940.00	1015.00	940.00	1015.00	940.00	980.00	1005.00	1020.00	1005.00	1050.00
PLS weight (g)	964.30	974.40	916.80	955.50	1001.30	1087.70	985.20	1061.30	1008.70	1049.70	1053.70	1066.20	1034.00	1070.20
Wash water volume (ml)	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00
Wash water weight (g)	998.23	998.23	998.23	998.23	998.23	998.23	998.23	998.23	998.23	998.23	998.23	998.23	998.23	998.23
Wash filtrate volume (ml)	1000.00	1070.00	1060.00	1085.00	1085.00	1065.00	1080.00	1000.00	1005.00	990.00	945.00	960.00	980.00	980.00
Wash filtrate weight (g)	987.00	1050.90	1051.40	1056.90	1084.60	1061.40	1059.70	993.60	985.90	973.50	935.60	949.30	976.00	977.40
Filter paper weight (g)	3.65	3.66	3.61	3.59	3.63	3.68	3.66	3.62	3.64	3.64	3.55	3.62	3.60	3.59
Wet cake + filter paper (g)	490.40	486.80	530.80	532.70	525.90	476.60	476.80	470.40	444.30	433.60	466.30	431.40	453.50	452.70
Dry cake weight (g)	382.80	379.80	367.20	368.80	352.30	352.70	360.30	360.10	334.80	334.00	355.20	352.50	375.20	376.10

P_{80}	212													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Feed solids (S-samples)														
T-Cu, %	0.9743	0.8798	1.6351	1.6712	3.7709	3.8785	3.8058	3.9095	7.38	7.7608	4.7625	4.9555	1.7581	1.7886
OX-Cu, %	0.6626	0.7397	1.5797	1.5381	2.7293	2.7476	3.7747	3.6373	7.00	7.0871	4.7435	4.6541	1.5542	1.5076
% OX-Cu	68%	84%	97%	92%	72%	71%	99%	93%	95%	91%	100%	94%	88%	84%
T-Co, %	0.0939	0.1891	0.2029	0.2658	0.1108	0.1415	0.6247	0.6477	0.28	0.3501	0.2642	0.2642	0.1798	0.2151
OX-Co, %	0.0173	0.0541	0.0982	0.1325	0.0418	0.0541	0.5761	0.6129	0.12	0.2747	0.2256	0.1497	0.1497	0.1202
Fe, %	1.2057	1.3084	0.6042	0.6941	0.4869	0.4942	1.1452	1.0297	0.87	0.618	0.3063	0.2925	0.1706	0.1687
Leach residue (C-samples)														
T-Cu, %	0.2943	0.3019	0.326	0.2958	1.3992	1.4419	0.4803	0.4407	0.5	0.5214	0.3751	0.3675	0.2958	0.3278
OX-Cu, %	0.0832	0.0863	0.3217	0.2374	0.4346	0.4422	0.2458	0.1992	0.4544	0.4559	0.129	0.3183	0.0817	0.0748
% OX-Cu	28%	29%	99%	80%	31%	31%	51%	45%	91%	87%	34%	87%	28%	23%
T-Co, %	0.1016	0.1016	0.163	0.117	0.3302	0.0648	0.7183	0.5443	0.2228	0.2243	0.1952	0.5618	0.1553	0.1844
OX-Co, %	0.0002	0.0051	0.0541	0.0369	0.0146	0.0247	0.505	0.5312	0.1006	0.1717	0.0933	0.1031	0.1178	0.0859
Fe, %	1.4789	1.556	0.673	0.6904	0.475	0.4851	1.2461	1.2534	0.7821	0.7574	0.3466	0.2824	0.1614	0.1697
PLS (PLS-samples)														
pH	1.59	1.72	1.57	1.8	1.68	1.61	1.8	1.74	1.51	1.56	1.55	1.61	1.76	1.71
Free Acid g/L	4.2	3.01	4.59	2.61	3.8	4.59	3.01	3.01	5.39	4.99	5.39	3.8	2.61	3.01
Cu (g/L)	3.2159	2.9599	6.192	5.5438	9.3986	9.8605	14.1005	14.6859	27.4753	26.4994	18.8242	21.2979	6.4258	6.9335
Co (g/L)	0.2697	0.2831	0.8876	0.5132	0.3736	0.3802	0.5948	0.6086	1.3819	1.3231	0.5478	0.5913	0.4332	0.5123
Fe (g/L)	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
Wash water (WW-samples)														
pH	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
Free Acid g/L	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
Cu (g/L)	0.4204	0.4349	1.3184	1.0518	2.0105	1.6989	2.3412	1.7906	3.4422	2.2585	1.8344	1.3855	0.5121	0.3626
Co (g/L)	0.0779	0.0849	0.1713	0.6622	0.1416	0.1406	0.3767	0.1211	0.1951	0.1379	0.1117	0.0868	0.0845	0.0816
Fe (g/L)	0.006	0.006	0.03	0.03	0.05	0.03	0.001	0.01	0.008	0.005	0.01	0.003	0.006	0.003
Cu Accountability	**118%**	**126%**	**122%**	**109%**	**106%**	**109%**	**115%**	**117%**	**105%**	**96%**	**115%**	**123%**	**115%**	**124%**
Leach Efficiencies														
T-Cu recovery (solids)	**71%**	**67%**	**82%**	**84%**	**67%**	**67%**	**89%**	**90%**	**94%**	**94%**	**93%**	**93%**	**84%**	**83%**
OX-Cu recovery (solids)	**88%**	**89%**	**81%**	**86%**	**86%**	**86%**	**94%**	**95%**	**95%**	**95%**	**98%**	**94%**	**95%**	**95%**
T-Co recovery (solids)	**-4%**	**49%**	**26%**	**59%**	**-162%**	**60%**	**-4%**	**24%**	**34%**	**47%**	**34%**	**-87%**	**19%**	**19%**

P_{80}	212													
Ore ID	RATLILLAS		RATGR		DSTRAT		SDS		SDB		RSF		RSC	
OX-Co recovery (solids)	99%	91%	49%	74%	69%	60%	21%	22%	30%	48%	63%	39%	26%	33%
Fe recovery (solids)	-17%	-13%	-2%	8%	14%	13%	2%	-10%	25%	-2%	0%	15%	11%	5%
Solids mass reduction, %	4.3%	5.1%	8.2%	7.8%	11.9%	11.8%	9.9%	10.0%	16.4%	16.5%	11.2%	11.9%	6.3%	6.0%
TAC (kg/t - ore))	97.92	92.86	217.24	203.62	291.23	296.87	156.68	153.41	147.46	147.63	109.27	109.27	97.39	99.69
GAC (kg/t-ore) [Cu Solids]	87.23	83.70	196.62	182.03	252.05	256.63	104.62	99.19	40.00	34.56	40.90	37.78	74.54	76.84
FAC (kg/t-ore) [Cu Solids]	90.87	86.81	203.63	189.37	265.37	270.31	122.32	117.63	76.54	73.00	64.14	62.08	82.31	84.61

Table 26: Oliveira 150µm leach test results

P_{80}	150													
Ore ID	RATLILLAS		RATGR		DSTRAT		SDS		SDB		RSF		RSC	
Test ID	WOL- T - 01		WOL-T-14		WOL-T-11		WOL-T-15		WOL-T12		WOL-T-13		WOL-T-07	
Samples	A	B	A	B	A	B	A	B	A	B	A	B	A	B
Time (minutes)	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH
0	1.420	1.50	1.499	1.491	1.401	1.41	1.337	1.46	1.478	1.377	1.357	1.50	1.491	1.48
10	1.475	1.50	1.530	1.483	1.839	1.90	1.557	1.52	1.926	1.499	1.446	1.47	1.458	1.48
20	1.512	1.49	1.453	1.492			1.526	1.51			1.475	1.49	1.481	1.53
30	1.487	1.50	1.496	1.442	1.516	1.56	1.517	1.49	1.493	1.511	1.460	1.49	1.460	1.51
60	1.490	1.49	1.493	1.524	1.474	1.47	1.499	1.51	1.469	1.401	1.493	1.51	1.469	1.41
90	1.474	1.50	1.478	1.450	1.429	1.48	1.531	1.50	1.483	1.503	1.494	1.46	1.503	1.52
120	1.503	1.47	1.486	1.270	1.488	1.45	1.473	1.47	1.461	1.500	1.457	1.52	1.155	1.47
150	1.509	1.51	1.431	1.415	1.485	1.50	1.501	1.50	1.481	1.491	1.485	1.50	1.232	1.34
180	1.502	1.50	1.504	1.498	1.271	1.48	1.498	1.50	1.431	1.504	1.482	1.50	1.254	1.43
210	1.497	1.50	1.504	1.499	1.391	1.47	1.490	1.49	1.405	1.533	1.448	1.48	1.264	1.45
240	1.502	1.50	1.494	1.498	1.442	1.48	1.507	1.50	1.421	1.497	1.478	1.50	1.271	1.46
Average	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.4	1.5
Time (minutes)	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP
0			319.3	327.4	326.5	303.1	330.6	301.1	325.4	335.1	323.1	295.1	317.7	296.8
10	318.4	335.2	319.7	328.0	300.8	276.6	320.5	301.7	298.1	327.7	325.0	301.2	320.3	298.4
20	319.6	336.8	324.7	326.8			322.0	300.5			323.2	299.6	319.0	295.1
30	321.7	364.0	321.2	328.6	322.1	296.7	321.7	302.1	322.6	324.9	323.2	298.3	320.2	295.6
60	321.2	302.9	319.0	329.2	323.1	299.7	319.7	296.4	320.1	327.5	318.3	294.6	317.9	299.0
90	322.1	307.5	327.3	328.6	322.1	295.8	315.3	296.3	321.6	328.8	315.5	308.6	322.1	296.8
120	319.4	306.0	322.3	335.6	316.1	295.3	319.3	298.7	319.5	324.5	321.7	299.3	356.1	314.9

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
150	321.0	300.7	322.4	329.7	317.5	311.3	318.0	296.2	320.0	324.6	326.2	297.1	340.8	310.5
180	321.7	304.4	320.0	325.9	327.3	303.5	318.0	296.1	330.3	330.5	321.2	294.3	333.8	302.7
210	320.5	304.4	325.2	325.8	325.3	297.9	321.8	298.6	325.4	325.5	320.4	297.0	329.5	297.5
240	319.1	303.1	322.3	324.3	321.2	294.9	317.2	294.8	321.6	326.1	316.4	293.7	326.6	294.2
Average	**320**	**317**	**322**	**328**	**320**	**297**	**320**	**298**	**320**	**328**	**321**	**298**	**328**	**300**
H2SO4 conc (g/l)	501.76	501.76	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82
Sample weight (g)	400.00	400.00	400.70	400.12	400.14	400.12	400.00	400.00	400.60	400.18	400.11	400.10	400.00	400.00
Acid to pH @ t=0 (ml)	7.00	7.00	4.90	5.90	5.80	4.90	6.00	4.50	5.50	8.50	6.10	4.00	3.90	3.90
Total Leach acid (ml)	61.50	64.10	93.10	92.50	151.00	154.20	137.20	141.70	120.80	116.60	72.80	74.90	84.00	79.90
PLS volume (ml)	810.00	805.00	855.00	940.00	930.00	1013.00	940.00	950.00	948.00	980.00	950.00	990.00	965.00	1016.00
PLS weight (g)	821.10	818.50	871.60	964.00	980.50	1064.50	992.50	1007.40	1000.10	1040.80	974.80	1008.20	1000.70	1051.70
Wash water volume (ml)	1100.00	1100.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00
Wash water weight (g)							980.00	980.00					998.25	998.25
Wash filtrate volume (ml)	1100.00	1095.00	1200.00	1000.00	1080.00	1020.00	1100.00	1000.00	1080.00	1010.00	1200.00	990.00	1050.00	970.00
Wash filtrate weight (g)	1086.60	1086.70	1049.60	991.50	1081.10	1020.30	1099.90	1002.30	1073.90	1050.00	1004.10	979.60	1051.70	970.50
Filter paper weight (g)	3.53	3.61	3.57	3.55	3.56	3.64	3.61	3.64	3.58	3.56	3.64	3.54	3.57	3.56
Wet cake + filter paper (g)	573.20	559.70	558.70	557.50	580.70	541.60	495.80	526.30	493.40	498.70	485.60	485.30	467.30	446.80
Dry cake weight (g)	382.83	389.10	374.78	374.42	356.18	357.16	330.63	327.82	350.12	349.11	364.37	367.81	375.51	375.61
Feed solids (S-samples)														
T-Cu, %	0.1433	0.1344	1.456	1.4395	2.9756	3.2369	2.3448	2.4533	5.221	5.3456	2.903	2.8627	1.6526	1.57
OX-Cu, %	0.1409	0.13	1.4179	1.3663	2.932	3.1068	2.33	2.3622	5.1429	5.1818	2.898	2.7954	1.4697	1.5222
% OX-Cu	98%	97%	97%	95%	99%	96%	99%	96%	99%	97%	100%	98%	89%	97%
T-Co, %	0.0918	0.1854	0.2016	0.2076	0.2847	0.2128	0.4023	0.377	0.922	0.8987	0.1625	0.1698	0.3413	0.27
OX-Co, %	0.0922	0.0802	0.1916	0.2092	0.2212	0.2039	0.3205	0.3173	0.8172	0.8713	0.1393	0.1381	0.3127	0.2721
Fe, %	1.4363	1.4287	0.6224	0.5815	0.382	0.4006	1.0579	1.0455	0.6699	0.6526	0.7	0.6511	0.3777	0.396
Leach residue (C-samples)														
T-Cu, %	0.0791	0.0377	0.0836	0.0905	0.1983	0.1656	0.4333	0.5434	0.2385	0.2552	0.1539	0.1465	0.1133	0.098
OX-Cu, %	0.0658	0.0356	0.0827	0.0909	0.1213	0.1376	0.1806	0.1884	0.2221	0.2328	0.1502	0.1418	0.0798	0.0886
% OX-Cu	83%	94%	99%	100%	61%	83%	42%	35%	93%	91%	98%	97%	70%	90%
T-Co, %	0.1092	0.123	0.1246	0.1195	0.1288	0.0646	0.1093	0.1014	0.658	0.8118	0.0715	0.0714	0.1638	0.07
OX-Co, %	0.09	0.0742	0.1079	0.1216	0.0861	0.0637	0.0781	0.0801	0.6097	0.8046	0.0728	0.0746	0.0962	0.0657

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Fe, %	1.4694	1.4685	0.8096	0.6999	0.3564	0.5033	0.7846	0.7422	0.5322	0.7064	0.8623	0.6479	0.2784	0.318
PLS (PLS-samples)														
pH	1.56	1.48	1.54	1.53	1.62	1.54	1.6	1.56	1.48	1.49	1.63	1.58	1.88	1.53
Free Acid g/L	3.76	3.78	4.16	2.46	2.98	4.16	3.37	2.59	4.94	4.94	3.35	3.76	1.02	4.43
Cu (g/L)	0.602	0.4262	4.6566	4.4374	9.8993	9.6485	7.8405	8.9859	17.6159	17.5778	9.5609	9.5647	4.8465	5.21
Co (g/L)	0.1612	0.127	0.5484	0.4874	0.6349	0.7128	0.4713	0.5714	0.8041	0.7432	0.5727	0.5605	0.8547	0.92
Fe (g/L)	0.04	0.05	0.1292	0.1401	0.1633	0.1608	0.23	0.26	0.1803	0.1693	0.1596	0.1608	0.44	0.36
Wash water (WW-samples)														
pH	2.24	2.07	1.91	2.13	1.74	1.91	2.28	2.24	1.98	2.04	1.95	1.98	1.68	1.86
Free Acid g/L	0.01	0.01	0.01	0.01	4.51	0.01	0.01	0.01	0.01	0.01	0.01	0.01	2.2	0.01
Cu (g/L)	0.08	0.07	1.1094	1.1075	2.1784	1.7841	1.3029	1.2479	2.7181	3.8007	1.502	1.2657	0.6777	0.27
Co (g/L)	0.02	0.02	0.2438	0.2499	0.1172	0.2317	0.0928	0.0781	0.2317	0.256	0.1708	0.1708	0.127	0.05
Fe (g/L)	0.00	0.01	0.0458	0.0472	0.0382	0.0444	0.0177	0.0421	0.0378	0.043	0.0396	0.0359	0.0372	0.02
Cu Accountability	**153 %**	**105 %**	**96%**	**98%**	**103 %**	**94%**	**109 %**	**118 %**	**98%**	**103 %**	**99%**	**98%**	**88%**	**94%**
Leach Efficiencies														
T-Cu recovery (solids)	47%	73%	95%	94%	94%	95%	85%	82%	96%	96%	95%	95%	94%	94%
OX-Cu recovery (solids)	55%	73%	95%	94%	96%	96%	94%	93%	96%	96%	95%	95%	95%	95%
T-Co recovery (solids)	-14%	35%	42%	46%	60%	73%	78%	78%	38%	21%	60%	61%	55%	76%
OX-Co recovery (solids)	7%	10%	47%	46%	65%	72%	80%	79%	35%	19%	52%	50%	71%	77%
Fe recovery (solids)	2%	0%	-22%	-13%	17%	-12%	39%	42%	31%	6%	-12%	9%	31%	25%
Solids mass reduction, %	4.3%	2.7%	6.5%	6.4%	11.0%	10.7%	17.3%	18.0%	12.6%	12.8%	8.9%	8.1%	6.1%	6.1%
TAC (kg/t - ore))	78.28	81.55	138.53	142.44	218.03	229.14	207.87	214.15	178.34	176.32	110.89	109.55	124.51	115.01
GAC (kg/t-ore) [Cu Solids]	77.24	80.04	117.27	121.53	174.82	181.46	177.20	183.16	100.97	97.26	68.25	67.44	100.64	92.20
FAC (kg/t-ore) [Cu Solids]	77.60	80.56	124.50	128.64	189.51	197.68	187.63	193.70	127.28	124.14	82.75	81.76	108.76	99.96

Table 27: Oliveira 212µm leach test results

P_{80}	212													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Test ID	**WOL - T - 09**		**WOL - T - 06**		**WOL - T - 10**		**WOL - T - 07**		**WOL - T - 12**		**WOL - T - 08**		**WOL - T - 13**	
Samples	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**

P_{80}	212													
Ore ID	RATLILLAS		RATGR		DSTRAT		SDS		SDB		RSF		RSC	
Time (minutes)	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH
0	1.502	1.425	1.413	1.314	1.520	1.510	1.450	1.470	1.480	1.430	1.496	1.447	1.507	1.425
10	1.387	1.161	2.141	2.318	2.300	1.960	1.480	1.520	1.750	1.490	1.631	1.737	1.424	1.557
20	1.398	1.538	1.437	1.551	1.580	1.450	1.450	1.520	1.510	1.500	1.658	1.683	1.414	1.515
30	1.347	1.465	1.557	1.534	1.900	1.640	1.500	1.450	1.520	1.510	1.570	1.504	1.502	1.508
60	1.512	1.307	1.521	1.536	1.490	1.560	1.500	1.500	1.500	1.500	1.502	1.502	1.503	1.506
90	1.501	1.388	1.537	1.511	1.500	1.500	1.510	1.190			1.467	1.507		
120	1.500	1.471	1.535	1.417	1.500	1.500	1.520	1.480	1.500	1.510	1.292	1.503	1.470	1.464
150	1.506	1.500	1.502	1.498	1.500	1.500	1.480	1.470	1.490	1.450	1.481	1.473	1.447	1.457
180	1.507	1.504	1.508	1.511	1.500	1.500	1.480	1.480	1.480	1.490	1.473	1.490	1.481	1.491
210	1.498	1.503	1.506	1.501	1.500	1.500	1.500	1.500	1.490	1.470	1.491	1.501	1.500	1.487
240	1.505	1.505	1.500	1.502	1.490	1.500	1.490	1.500	1.500	1.490	1.501	1.504	1.496	1.502
Average	1.469	1.433	1.560	1.563	1.616	1.556	1.487	1.462	1.522	1.484	1.506	1.532	1.474	1.491
Time (minutes)	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP
0	324.0	328.8	328.4	335.0	324.1	315.2	327.5	317.1	323.1	317.0	315.2	322.6	318.5	324.2
10	333.7	360.2	285.0	280.2	285.5	294.6	327.5	316.1	313.4	319.0	314.8	314.2	327.4	320.8
20	332.8	336.2	331.6	326.9	329.4	324.9	329.7	316.8	328.2	320.0	313.6	317.3	328.0	325.5
30	336.0	339.4	323.8	328.5	309.5	314.0	326.2	320.2	328.1	319.8	318.7	327.3	321.9	325.6
60	322.5	340.7	325.0	327.7	330.1	318.3	326.5	317.2	328.0	318.6	321.9	325.8	321.5	326.2
90	322.8	333.0	323.1	327.3	329.5	318.4	325.7	339.3			323.0	324.3		
120	321.7	329.5	321.4	331.0	328.9	321.4	324.4	322.5	324.6	316.9	332.8	323.5	324.8	327.7
150	321.7	325.8	321.7	325.0	326.6	319.1	325.7	321.0	325.7	320.1	323.0	324.6	324.5	327.5
180	323.9	325.7	322.6	326.8	325.1	319.1	324.9	319.2	325.4	316.8	322.8	323.4	321.0	324.0
210	321.9	326.2	321.6	327.1	325.2	318.4	325.5	317.2	329.6	318.9	323.3	326.4	321.9	328.1
240	322.5	324.6	321.0	325.8	325.9	318.2	325.0	316.2	326.9	316.8	321.7	325.1	320.4	324.3
Average	325.8	333.6	320.5	323.8	321.8	316.5	326.2	320.3	325.3	318.4	321.0	323.1	323.0	325.4
H2SO4 conc (g/l)	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79
Sample weight (g)	400.19	400.05	400.10	400.13	400.23	400.20	400.04	400.16	400.03	400.12	400.39	400.02	400.36	400.26
Acid to pH @ t=0 (ml)	5.10	6.10	5.90	7.10	4.00	3.00	4.90	3.20	4.20	3.10	4.50	5.00	4.00	5.90
Total Leach acid (ml)	45.90	47.60	112.60	110.90	209.00	208.30	125.00	120.50	116.90	111.90	89.30	90.30	67.10	66.40
PLS volume (ml)	930.00	1065.00	930.00	915.00	1080.00	1060.00	1040.00	970.00	1010.00	990.00	1020.00	980.00	1020.00	1020.00
PLS weight (g)	935.40	1072.80	960.10	944.80	1161.00	1130.70	1085.40	1004.60	1070.10	1050.20	1064.80	1020.60	1050.70	1045.20

P_{80}	212													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Wash water volume (ml)	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00
Wash water weight (g)	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25
Wash filtrate volume (ml)	1000.00	1000.00	1155.00	1110.00	990.00	975.00	980.00	1010.00	1000.00	1040.00	1000.00	1010.00	1010.00	1030.00
Wash filtrate weight (g)	995.20	981.10	1137.70	1123.90	982.20	970.00	965.50	1006.60	993.00	1022.00	995.20	1004.10	995.30	1009.20
Filter paper weight (g)	3.66	3.66	3.70	3.68	3.62	3.69	3.66	3.68	3.56	3.58	3.68	3.62	3.62	3.61
Wet cake + filter paper (g)	514.90	501.10	508.04	526.30	531.70	553.90	496.50	590.20	471.00	483.60	461.40	477.90	457.10	451.20
Dry cake weight (g)	388.00	382.20	377.90	375.90	352.70	361.00	368.00	367.40	349.40	349.90	364.50	363.30	377.30	376.50
Feed solids (S-samples)														
T-Cu, %	0.171	0.203	0.76	0.82	3.336	3.35	2.439	2.495	5.507	5.6683	3.52	3.544	1.3606	1.3184
OX-Cu, %	0.162	0.181	0.75	0.71	3.192	3.336	2.166	2.066	5.3796	5.1481	3.31	3.452	1.2354	1.3137
% OX-Cu	95%	89%	99%	87%	96%	100%	89%	83%	98%	91%	94%	97%	91%	100%
T-Co, %	0.099	0.08	0.28	0.26	0.382	0.365	0.345	0.485	0.9925	1.0005	0.201	0.222	0.3101	0.346
OX-Co, %	0.007	0.033	0.16	0.14	0.223	0.297	0.232	0.272	0.932	0.9275	0.105	0.105	0.2502	0.3071
Fe, %	0.503	0.573	0.49	0.52	0.228	0.232	0.931	0.991	0.6425	0.6671	0.503	0.439	0.318	0.356
Leach residue (C-samples)														
T-Cu, %	0.01	0.029	0.12	0.12	0.429	0.332	0.255	0.134	0.1022	0.138	0.365	0.373	0.4083	0.39
OX-Cu, %	0.003	0.025	0.09	0.1	0.041	0.07	0.094	0.051	0.1022	0.1371	0.057	0.075	0.3654	0.3
% OX-Cu	30%	86%	75%	83%	10%	21%	37%	38%	100%	99%	16%	20%	89%	77%
T-Co, %	0.079	0.03	0.16	0.18	0.088	0.129	0.221	0.248	0.1417	0.1348	0.11	0.084	0.97	0.8319
OX-Co, %	0.013	0.013	0.06	0.08	0.013	0.029	0.203	0.146	0.0999	0.1144	0.01	0.013	0.9	0.7961
Fe, %	0.894	0.744	0.54	0.5	0.215	0.195	1.08	1.057	0.2863	0.267	0.524	0.419	0.7002	0.7036
PLS (PLS-samples)														
pH	1.47	1.36	1.42	1.4	1.36	1.7	1.44	1.8	1.53	1.51	1.55	1.48	1.42	1.77
Free Acid g/L	0.1	0.12	4.75	5.14	0.12	0.09	0.1	0.09	1.75	2.14	0.09	0.1	2.54	0.56
Cu (g/L)	0.699	0.5965	2.6658	2.7605	10.6099	9.4816	7.7019	7.1457	4.3173	4.2248	11.7065	11.6931	17.737	17.2136
Co (g/L)	0.3319	0.3026	0.6488	0.6049	1.1127	0.9614	0.6833	0.5857	0.8663	0.9157	0.654	0.6686	1.1167	1.0797
Fe (g/L)	0.0176	0.0018	0.0875	0.0748	0.1395	0.0956	0.241	0.1454	0.1695	0.1931	0.1337	0.1473	0.0403	0.0009
Wash water (WW-samples)														
pH	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
Free Acid g/L	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
Cu (g/L)	0.172	0.0781	0.5048	0.627	1.1771	1.7931	0.699	1.5931	0.5296	0.3604	1.1735	1.5162	1.8672	2.2604
Co (g/L)	0.1367	0.122	0.1317	0.1512	0.2294	0.2831	0.1562	0.205	0.2838	0.2394	0.1464	0.1611	0.2196	0.2591
Fe (g/L)	0.01	0.01	0.0108	0.029	0.01	0.03	0.01	0.01	0.0024	0.0033	0.03	0.02	0.003	0.0037

P_{80}	212													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Cu Accountability	**126 %**	**101 %**	**116 %**	**112 %**	**106 %**	**97%**	**99%**	**90%**	**24%**	**22%**	**102 %**	**101 %**	**395 %**	**405 %**
Leach Efficiencies														
T-Cu recovery (solids)	94%	86%	85%	86%	89%	91%	90%	95%	98%	98%	91%	90%	72%	72%
OX-Cu recovery (solids)	98%	87%	89%	87%	99%	98%	96%	98%	98%	98%	98%	98%	72%	79%
T-Co recovery (solids)	23%	64%	46%	35%	80%	68%	41%	53%	88%	88%	50%	66%	-195 %	-126 %
OX-Co recovery (solids)	-80%	62%	65%	46%	95%	91%	20%	51%	91%	89%	91%	89%	-239 %	-144 %
Fe recovery (solids)	-72%	-24%	-4%	10%	17%	24%	-7%	2%	61%	65%	5%	13%	-108 %	-86%
Solids mass reduction, %	3.0%	4.5%	5.5%	6.1%	11.9 %	9.8%	8.0%	8.2%	12.7 %	12.6 %	9.0%	9.2%	5.8%	5.9%
TAC (kg/t - ore))	67.76	70.30	166.27	163.74	308.51	307.50	184.60	177.90	172.65	165.22	131.77	133.36	99.02	98.01
GAC (kg/t-ore) [Cu Solids]	65.27	67.59	156.29	152.83	262.86	260.42	150.58	141.29	89.03	79.60	82.57	83.89	83.95	83.32
FAC (kg/t-ore) [Cu Solids]	66.12	68.51	159.68	156.54	278.38	276.43	162.15	153.74	117.46	108.71	99.29	100.71	89.08	88.31

Table 28: Virgule 150µm leach test results

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Test ID	**WOL - T -02**		**WOL - T -16**		**WOL - T -17**		**WOL - T -21**		**WOL - T -18**		**WOL - T -19**		**WOL - T -20**	
Samples	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**
Time (minutes)	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**
0	1.563	1.453	1.362	1.460	1.510	1.48	1.505	1.41	1.510	1.496	1.547	1.46	1.542	1.577
10	1.498	1.498	1.734	1.650	1.530	1.46	1.457	1.38	1.679	1.679	1.713	1.50	1.604	1.497
20	1.499	1.505	1.517	1.516	1.507	1.44	1.410	1.48	1.476	1.474	1.510	1.52	1.430	1.501
30	1.508	1.501	1.523	1.504	1.493	1.47	1.443	1.46	1.476	1.508	1.512	1.51	1.418	1.441
60	1.500	1.502	1.495	1.502	1.500	1.51	1.477	1.52	1.501	1.476	1.373	1.40	1.426	1.504
90	1.504	1.509	1.510	1.503	1.513	1.52	1.505	1.50	1.498	1.501	1.379	1.45	1.470	1.506
120	1.505	1.466	1.501	1.504	1.500	1.51	1.500	1.50	1.505	1.503	1.393	1.46	1.492	1.499
150	1.505	1.481	1.488	1.508	1.504	1.51	1.492	1.50	1.498	1.498	1.408	1.47	1.508	1.509
180	1.495	1.495	1.503	1.498	1.503	1.51	1.497	1.49	1.492	1.493	1.407	1.47	1.483	1.490
210	1.499	1.496	1.514	1.504	1.511	1.51	1.506	1.49	1.505	1.503	1.393	1.46	1.474	1.492

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
240	1.501	1.502	1.491	1.498	1.492	1.50	1.479	1.49	1.487	1.468	1.376	1.45	1.468	1.491
Average	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**
Time (minutes)	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**
0	332.3	330.2	329.3	331.9	317.7	296.4	319.3	298.8	309.2	340	324.5	294.1	325.7	325.9
10	331.5	331.3	309.3	320.6	324.4	301.0	327.8	306.8	305.1	305.1	319.7	299.4	322.2	332.7
20	330.3	330.2	320.3	327.4	330.1	302.6	330.8	304.0	319.4	336.5	330.6	299.9	332.1	332.4
30	331.5	329.3	318.5	325.8	328.8	300.4	328.4	305.8	320.2	332.0	328.8	301.1	331.3	334.9
60	329.2	330.1	323.4	327.2	322.3	294.8	324.8	300.1	320.2	327.8	333.2	305.1	329.3	329.7
90	330.9	333.5	319.3	326.8	318.3	295.3	324.2	311.2	319.7	328.8	331.8	305.8	327.0	330.6
120	333.2	331.3	318.7	327.2	318.1	296.4	325.9	308.2	320.1	326.6	332.3	305.3	326.5	328.8
150	331.6	330.7	321.8	328.8	317.5	295.5	325.1	304.9	321.1	327.3	333.0	305.1	327.1	328.4
180	330.3	329.8	321.8	325.5	316.0	297.7	323.5	303.4	319.7	327.0	332.7	304.3	327.0	332.8
210	330.2	330.0	320.9	330.2	317.2	295.7	326.5	301.9	321.1	329.2	330.8	302.7	324.8	330.2
240	331.5	330.0	320.5	326.3	316.5	295.6	326.7	300.7	320.2	327.0	329.9	301.8	323.5	328.6
Average	**331**	**331**	**320**	**327**	**321**	**297**	**326**	**304**	**318**	**328**	**330**	**302**	**327**	**330**
H2SO4 conc (g/l)	501.76	501.76	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82
Sample weight (g)	400.00	400.00	400.00	400.00	400.17	400.05	400.07	400.07	400.06	400.20	400.06	400.10	400.15	400.18
Acid to pH @ t=0 (ml)	7.00	7.00	6.50	5.50	4.10	4.50	4.50	6.30	4.30	5.60	4.20	5.30	5.10	5.30
Total Leach acid (ml)	21.60	17.50	82.10	82.70	121.10	120.70	20.80	20.10	86.80	87.70	77.60	77.10	37.30	37.90
PLS volume (ml)	820.00	870.00	1020.00	920.00	1040.00	1060.00	945.00	935.00	1035.00	1000.00	1045.00	1030.00	1025.00	990.00
PLS weight (g)	830.20	878.90	1086.90	985.90	1123.30	1146.30	958.20	948.00	1109.90	1072.00	1104.10	1094.40	1054.20	1024.20
Wash water volume (ml)	1135.00	1085.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00
Wash water weight (g)			978.50	979.10									980.40	981.20
Wash filtrate volume (ml)	1135.00	1085.00	980.00	940.00	981.00	980.00	980.00	1000.00	1015.00	1020.00	1005.00	1000.00	970.00	980.00
Wash filtrate weight (g)	1029.20	1067.70	976.00	937.00	983.00	988.80	965.30	996.00	1008.30	1015.30	994.50	989.60	962.00	969.90
Filter paper weight (g)	3.52	3.57	3.59	3.60	3.59	3.61	3.62	3.58	3.55	3.56	3.62	3.55	3.55	3.56
Wet cake + filter paper (g)	496.60	487.90	436.60	447.40	453.70		518.30	531.60	421.80	408.30	414.20	410.50	444.20	437.70
Dry cake weight (g)	384.66	378.98	363.57	365.78	343.86	342.60	384.13	386.61	331.43	334.51	345.46	343.66	372.29	369.21
Feed solids (S-samples)														
T-Cu, %	0.6938	0.7279	7.38	7.48	8.95	8.63	1.0678	1.0533	7.9	8.34	6.9883	6.8552	2.5198	2.6571
OX-Cu, %	0.7089	0.7234	7	7.02	8.18	7.63	1.0038	1.0059	7.24	7.57	6.6382	6.8073	2.5124	2.488
% OX-Cu	102%	99%	95%	94%	91%	88%	94%	95%	92%	91%	95%	99%	100%	94%

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
T-Co, %	0.1445	0.1777	0.28	0.28	0.29	0.26	0.4256	0.4311	0.77	0.79	0.1317	0.1684	0.4977	0.3841
OX-Co, %	0.1041	0.1179	0.12	0.22	0.12	0.13	0.4338	0.3887	0.72	0.73	0.0586	0.0635	0.4481	0.3076
Fe, %	2.7143	2.4589	0.87	0.88	0.61	0.66	3.6481	3.469	0.79	0.81	0.4102	0.4056	0.3497	0.3122
Leach residue (C-samples)														
T-Cu, %	0.0615	0.0874	0.22	0.23	0.26	0.26	0.2478	0.2399	0.34	0.31	0.465	0.4787	0.1631	0.1783
OX-Cu, %	0.061132	0.0676	0.19	0.21	0.18	0.2	0.1814	0.1851	0.32	0.3	0.2575	0.1616	0.0964	0.1607
% OX-Cu	99%	77%	86%	91%	69%	77%	73%	77%	94%	97%	55%	34%	59%	90%
T-Co, %	0.0975	0.1755	0.31	0.38	0.26	0.25	0.4042	0.4391	0.7	0.76	0.0322	0.0393	0.2816	0.2847
OX-Co, %	0.1018	0.1179	0.2	0.33	0.18	0.16	0.3468	0.3524	0.67	0.74	0.0031	0.0169	0.1807	0.1904
Fe, %	2.3931	2.6307	1.12	1.1	0.99	1.03	3.6844	3.7402	0.89	0.96	0.4083	0.4303	0.2682	0.303
PLS (PLS-samples)														
pH	1.49	1.34	1.53	1.49	1.7	1.58	1.53	1.46	1.54	1.43	1.45	1.39	1.46	1.46
Free Acid g/L	3.76	4.94	2.58	2.97	1.01	3.75	3.76	2.98	1.79	3.12	1.02	2.59	3.37	4.16
Cu (g/L)	2.7828	2.6302	25.32	28.68	30.9	30.04	3.2224	3.07 59	28.88	29.97	23.3977	24.583	9.4059	9.2406
Co (g/L)	0.2051	0.1294	0.12	0.16	0.57	0.73	0.1905	0.1832	0.42	0.39	0.6248	0.6615	0.6178	0.6515
Fe (g/L)	0.14	0.12	0.03	0.06	0.09	0.11	0.0673	0.0848	0.08	0.06	0.1646	0.1829	0.3695	0.1901
Wash water (WW-samples)														
pH	1.93	2.1	1.98	1.89	2.19	2.19	2.22	2.07	2.05	1.96	2.05	2.11	2.11	2.03
Free Acid g/L	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Cu (g/L)	0.4042	0.3382	2.6	2.82	4.44	3.13	0.464	0.6154	2.72	3.84	1.9442	1.1401	0.7558	0.699
Co (g/L)	0.0244	0.0293	0.02	0.09	0.1	0.00	0.0366	0.0464	0.00	0.02	0.1373	0.1373	0.0659	0.1422
Fe (g/L)	0.0157	0.0157	0.04	0.02	0.02	0.02	0.0157	0.0245	0.04	0.03	0.0312	0.0234	0.0255	0.0161
Cu Accountability	**107%**	**103%**	**99%**	**100%**	**104%**	**104%**	**104%**	**105%**	**107%**	**105%**	**100%**	**102%**	**109%**	**99%**
Leach Efficiencies														
T-Cu recovery (solids)	**91%**	**89%**	**97%**	**97%**	**98%**	**97%**	**78%**	**78%**	**96%**	**97%**	**94%**	**94%**	**94%**	**94%**
OX-Cu recovery (solids)	**92%**	**91%**	**98%**	**97%**	**98%**	**98%**	**83%**	**82%**	**96%**	**97%**	**97%**	**98%**	**96%**	**94%**
T-Co recovery (solids)	**35%**	**6%**	**-1%**	**-24%**	**23%**	**18%**	**9%**	**2%**	**25%**	**20%**	**79%**	**80%**	**47%**	**32%**
OX-Co recovery (solids)	**6%**	**5%**	**-51%**	**-37%**	**-29%**	**-5%**	**23%**	**12%**	**23%**	**15%**	**95%**	**77%**	**62%**	**43%**
Fe recovery (solids)	**15%**	**-1%**	**-17%**	**-14%**	**-39%**	**-34%**	**3%**	**-4%**	**7%**	**1%**	**14%**	**9%**	**29%**	**10%**

P_{80}	150													
Ore ID	RATLILLAS		RATGR		DSTRAT		SDS		SDB		RSF		RSC	
Solids mass reduction, %	3.8%	5.3%	9.1%	8.6%	14.1 %	14.4 %	4.0%	3.4%	17.2 %	16.4 %	13.6 %	14.1 %	7.0%	7.7%
TAC (kg/t - ore))	28.14	19.96	126.92	126.07	185.95	178.70	29.22	32.79	132.62	132.72	120.57	117.46	55.22	54.76
GAC (kg/t-ore) [Cu Solids]	18.34	10.00	16.10	13.86	51.26	48.94	16.41	20.11	15.03	7.99	18.91	18.00	18.67	16.29
FAC (kg/t-ore) [Cu Solids]	21.67	13.39	53.78	52.01	97.06	93.06	20.76	24.42	55.01	50.40	53.47	51.81	31.10	29.37

Table 29: Virgule 212µm leach test results

P_{80}	212													
Ore ID	RATLILLAS		RATGR		DSTRAT		SDS		SDB		RSF		RSC	
Test ID	WOL - T - 04		WOL - T - 05		WOL - T - 28		WOL - T - 02		WOL - T - 11		WOL - T - 03		WOL - T - 01	
Samples	A	B	A	B	A	B	A	B	A	B	A	B	A	B
Time (minutes)	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH
0	1.486	1.444	1.490	1.520	1.510	1.490	1.489	1.523	1.505	1.501	1.510	1.450	1.270	1.460
10	1.742	1.815	1.590	1.600	1.470	1.510	1.504	1.509	1.681	1.775	2.010	1.740	1.560	1.460
20	1.160	1.432	1.450	1.460	1.500	1.490	1.507	1.503	1.476	1.447	1.560	1.550	1.450	1.490
30	1.113	1.498	1.400	1.300	1.490	1.410	1.512	1.510	1.473	1.434	1.500	1.490	1.490	1.470
60	1.139	1.505	1.480	1.440	1.500	1.500	1.508	1.508	1.506	1.505	1.500	1.500	1.450	1.450
90	1.170	1.507	1.510	1.500	1.500	1.500	1.508	1.501	1.504	1.506	1.510	1.500	1.500	1.500
120	1.207	1.500	1.520	1.490	1.500	1.510	1.502	1.502	1.503	1.504	1.490	1.500	1.500	1.490
150	1.210	1.506	1.480	1.460	1.500	1.500	1.507	1.503	1.497	1.496	1.500	1.500	1.480	1.500
180	1.241	1.504	1.470	1.470	1.500	1.500	1.500	1.501	1.498	1.493	1.500	1.490	1.500	1.500
210	1.240	1.500	1.460	1.500	1.500	1.480	1.502	1.502	1.493	1.501	1.500	1.500	1.500	1.500
240	1.235	1.501	1.460	1.500	1.490	1.500	1.505	1.504	1.498	1.497	1.500	1.500	1.500	1.500
Average	1.268	1.519	1.483	1.476	1.496	1.490	1.504	1.506	1.512	1.514	1.553	1.520	1.473	1.484
Time (minutes)	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP
0	323.5	326.1	322.9	317.5			326.4	326.4	320.0	321.1	319.8	319.6	335.4	315.2
10	309.0	306.6	323.6	314.4	338.8	332.8	324.5	326.7	313.8	312.0	301.2	308.1	324.8	322.5
20	347.6	328.1	332.0	323.1	336.8	334.8	323.3	326.4	327.8	333.3	329.0	315.4	331.7	321.0
30	345.8	326.9	333.8	331.7	336.7	339.1	323.2	325.5	327.8	333.7	330.0	318.0	328.2	321.4
60	341.4	324.5	326.7	321.4	333.3	331.0	322.5	326.1	322.8	326.6	326.0	316.0	330.6	322.1
90	342.6	327.1	323.7	317.7	331.6	329.4	323.0	325.4	322.5	325.8	325.2	318.6	327.5	319.5
120	342.7	325.0	325.0	318.3	330.3	327.5	321.6	325.2	321.4	325.2	325.2	319.8	325.1	318.2
150	339.9	325.4	334.0	323.4	332.0	329.6	322.3	326.7	322.9	325.9	325.2	319.5	325.0	318.2
180	341.6	328.3	330.4	320.6	331.0	328.1	321.1	325.2	322.5	326.4	325.7	318.3	325.9	318.2
210	339.0	326.2	328.0	322.3	330.1	330.7	323.6	326.7	321.8	325.1	324.3	317.9	326.6	318.1

P_{80}	212													
Ore ID	RATLILLAS		RATGR		DSTRAT		SDS		SDB		RSF		RSC	
240	337.3	331.3	326.6	320.2	331.6	329.5	322.2	326.4	324.9	327.5	326.8	318.3	325.1	318.6
Average	337.3	325.0	327.9	321.0	333.2	331.3	323.1	326.1	322.6	325.7	323.5	317.2	327.8	319.4
H2SO4 conc (g/l)	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79
Sample weight (g)	400.01	400.02	400.01	400.38	400.35	400.35	400.01	400.01	400.10	400.44	400.10	400.01	400.12	400.52
Acid to pH @ t=0 (ml)	5.30	6.50	4.60	3.20	4.90	5.10	5.50	4.00	3.90	5.00	4.50	4.30	7.20	3.40
Total Leach acid (ml)	24.00	17.80	32.20	30.70	121.80	117.40	17.80	19.00	80.50	82.00	62.00	59.00	29.80	31.80
PLS volume (ml)	970.00	970.00	980.00	970.00	1030.00	1060.00	890.00	950.00	1000.00	980.00	980.00	1013.00	990.00	1010.00
PLS weight (g)	969.60	972.00	995.90	984.30	1106.60	1129.50	895.00	957.20	1064.40	1044.10	1029.40	1067.70	1010.50	1023.80
Wash water volume (ml)	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00
Wash water weight (g)	998.56	998.56	998.56	998.56	998.56	998.56	998.56	998.56	998.56	998.56	998.56	998.56	998.56	998.56
Wash filtrate volume (ml)	980.00	1020.00	1040.00	1060.00	980.00	1020.00	990.00	980.00	970.00	940.00	960.00	1005.00	960.00	1050.00
Wash filtrate weight (g)	970.70	1008.20	1023.30	1045.70	976.50	1014.80	971.40	964.40	962.40	931.10	947.70	988.30	949.60	1027.40
Filter paper weight (g)	3.71	3.72	3.66	3.74	3.64	3.63	3.67	3.70	3.70	3.64	3.66	3.70	3.67	3.67
Wet cake + filter paper (g)	488.90	483.80	466.70	454.10	430.40	432.10	510.10	496.50	455.90	422.10	464.70	451.90	434.30	446.00
Dry cake weight (g)	385.50	385.80	375.80	373.50	325.66	327.27	388.00	388.10	341.00	341.10	353.30	353.60	375.50	376.10
Feed solids (S-samples)														
T-Cu, %	1.09	0.96	3.34	2.47	8.6324	8.7112	0.6831	0.7248	7.7521	7.9935	6.63	6.54	2.5855	2.42
OX-Cu, %	0.96	0.92	2.47	2.32	8.1579	8.4133	0.62	0.7	7.6081	7.9728	6.08	6.41	2.3352	2.15
% OX-Cu	88%	96%	74%	94%	95%	97%	91%	97%	98%	100%	92%	98%	90%	89%
T-Co, %	0.13	0.16	0.16	0.19	0.1619	0.1423	0.079	0.0757	0.75	0.7982	0.18	0.21	0.4467	0.4231
OX-Co, %	0.08	0.07	0.11	0.09	0.1273	0.13	0.06	0.07	0.694	0.7168	0.12	0.1	0.3777	0.3869
Fe, %	1.58	1.3	0.88	0.87	0.6121	0.6189	3.1791	3.2038	0.7371	0.8408	0.29	0.3	0.2859	0.2413
Leach residue (C-samples)														
T-Cu, %	0.05	0.04	0.31	0.21	0.4626	0.5985	0.09	0.17	0.3828	0.2946	0.95	1.39	0.123	0.206
OX-Cu, %	0.05	0.02	0.26	0.16	0.1807	0.2827	0.0663	0.15	0.2349	0.2939	0.79	1.15	0.064	0.063
% OX-Cu	100%	50%	84%	76%	39%	47%	74%	88%	61%	100%	83%	83%	52%	30%
T-Co, %	0.13	0.08	0.12	0.12	0.0467	0.0662	0.0649	0.066	0.6611	0.738	0.08	0.11	0.342	0.323
OX-Co, %	0.06	0.01	0.05	0.07	0.0322	0.0461	0.06	0.06	0.6298	0.6778	0.03	0.02	0.296	0.295
Fe, %	1.67	1.54	0.94	0.93	0.7239	0.7487	4.6775	4.1548	0.9794	0.968	0.29	0.28	0.271	0.356
PLS (PLS-samples)														
pH	1.18	1.38	1.41	1.57	1.54	1.74	1.28	1.34	1.42	1.31	1.38	1.58	1.42	1.59
Free Acid g/L	6.32	3.97	4.28	2.79	4.94	2.59	6.1	4.94	2.54	3.33	4.72	2.79	1.94	0.95

P_{80}	212													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Cu (g/L)	3.7179	3.5709	10.1077	9.2611	29.2532	28.1979	2.87	2.83	27.5136	28.1526	23.6437	22.1523	9.53	8.42
Co (g/L)	0.2439	0.2268	0.2634	0.2415	0.5155	0.6135	0.18	0.16	0.9069	0.8946	0.6098	0.5756	0.76	0.60
Fe (g/L)	0.0728	0.0414	0.0522	0.012	0.1247	0.1174	0.14	0.12	0.0058	0.0046	0.0865	0.0802	0.10	0.06
Wash water (WW-samples)														
pH	NA	NA	NA	NA	NA	NA	1.93	2.1	NA	NA	NA	NA	NA	NA
Free Acid g/L	NA	NA	NA	NA	NA	NA	0.01	0.01	NA	NA	NA	NA	NA	NA
Cu (g/L)	0.4703	0.5031	1.0362	1.2868	3.0183	3.045	0.36	0.34	2.2214	2.1922	1.7784	1.7261	0.3434	1.1564
Co (g/L)	0.0146	0.0268	0.0293	0.0488	0.0885	0.0983	0.09	0.02	0.1999	0.1949	0.0537	0.039	0.2344	0.2443
Fe (g/L)	0.0067	0.0046	0.0051	0.0051	0.0196	0.0254	0.06	0.04	0.0039	0.0042	0.0079	0.0079	0.0038	0.0036
Cu Accountability	**98%**	**108 %**	**91%**	**113 %**	**100 %**	**100 %**	**119 %**	**127 %**	**100 %**	**96%**	**106 %**	**111 %**	**99%**	**108 %**
Leach Efficiencies														
T-Cu recovery (solids)	**96%**	**96%**	**91%**	**92%**	**96%**	**94%**	**87%**	**77%**	**96%**	**97%**	**87%**	**81%**	**96%**	**92%**
OX-Cu recovery (solids)	**95%**	**98%**	**90%**	**94%**	**98%**	**97%**	**90%**	**79%**	**97%**	**97%**	**89%**	**84%**	**97%**	**97%**
T-Co recovery (solids)	**4%**	**52%**	**30%**	**41%**	**77%**	**62%**	**20%**	**15%**	**25%**	**21%**	**61%**	**54%**	**28%**	**28%**
OX-Co recovery (solids)	**28%**	**86%**	**57%**	**27%**	**79%**	**71%**	**3%**	**17%**	**23%**	**19%**	**78%**	**82%**	**27%**	**28%**
Fe recovery (solids)	**-2%**	**-14%**	**0%**	**0%**	**4%**	**1%**	**-43%**	**-26%**	**-13%**	**2%**	**12%**	**17%**	**11%**	**-39%**
Solids mass reduction, %	**3.6%**	**3.6%**	**6.1%**	**6.7%**	**18.7 %**	**18.3 %**	**3.0%**	**3.0%**	**14.8 %**	**14.8 %**	**11.7 %**	**11.6 %**	**6.2%**	**6.1%**
TAC (kg/t - ore))	**35.45**	**26.29**	**47.56**	**45.30**	**179.74**	**173.25**	**26.29**	**28.06**	**118.87**	**120.98**	**91.55**	**87.14**	**44.00**	**46.91**
GAC (kg/t-ore) [Cu Solids]	**19.37**	**12.07**	**0.50**	**10.20**	**52.31**	**46.34**	**17.09**	**19.42**	**4.25**	**1.48**	**2.17**	**5.16**	**5.87**	**12.54**
FAC (kg/t-ore) [Cu Solids]	**24.83**	**16.90**	**16.50**	**22.13**	**95.64**	**89.49**	**20.22**	**22.36**	**43.22**	**42.11**	**32.56**	**33.04**	**18.84**	**24.23**

Table 30: Variante 150µm leach test results

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Test ID	**WOL - T -25***		**WOL - T -28**		**WOL - T -22**		**WOL - T -23**		**WOL - T -27**		**WOL - T -26***		**WOL - T -24**	
Samples	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**	**A**	**B**
Time (minutes)	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**	**pH**
0	1.496	1.530	1.487	1.515	1.539	1.631	1.317	1.35	1.524	1.54	1.424	1.49	1.394	1.48
10	1.497	1.505	1.532	1.521	1.548	2.228	1.506	1.46	1.479	1.51	1.517	1.49	1.440	1.49
20	1.505	1.508	1.509	1.508	1.381	3.295	1.421	1.52	1.499	1.51	1.477	1.50	1.427	1.41

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
30	1.502	1.498	1.505	1.501	1.573	1.603	1.469	1.46	1.500	1.50	1.480	1.50	1.428	1.47
60	1.501	1.499	1.504	1.503	1.488	1.518	1.370	1.39	1.500	1.49	1.494	1.49	1.469	1.47
90	1.502	1.502	1.501	1.500	1.521	1.515	1.495	1.43	1.502	1.50	1.495	1.51	1.505	1.51
120	1.502	1.494	1.505	1.505	1.488	1.493	1.507	1.51	1.504	1.50	1.495	1.50	1.502	1.50
150	1.501	1.477	1.496	1.495	1.464	1.498	1.508	1.50	1.499	1.50	1.500	1.50	1.505	1.50
180	1.499	1.471	1.503	1.506	1.468	1.498	1.505	1.50	1.495	1.50	1.495	1.50	1.505	1.50
210	1.511	1.496	1.506	1.502	1.484	1.495	1.494	1.50	1.510	1.50	1.482	1.48	1.503	1.51
240	1.494	1.498	1.497	1.499	1.498	1.496	1.496	1.49	1.504	1.50	1.469	1.48	1.495	1.50
Average	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.8**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**
Time (minutes)	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**	**ORP**
0	319.4	325	323.5	329.3	317.7	327.6	331.3	304.2	326.8	296.2	328.8	302.9	327.8	325.9
10	320.9	320.9	332.7	331.8	323.4	298.2	325.9	308.3	329.9	299.5	324.8	302.6	329.2	332.0
20	320.2	321.1	333.4	330.7	336.3	229.4	331.1	305.9	328.2	299.5	326.8	301.2	329.8	337.3
30	319.7	321.2	330.6	329.9	326.3	335.2	327.7	309.4	327.4	299.6	325.9	301.9	328.7	332.9
60	319.7	320.0	325.6	328.4	331.5	339.5	331.1	310.1	326.8	305.2	325.2	301.5	325.1	325.2
90	320.0	325.6	324.9	328.7	328.9	336.1	324.6	305.2	327.4	301.2	324.4	301.5	322.8	329.4
120	319.9	325.9	325.2	329.1	328.9	335.1	324.4	301.2	326.8	299.8	325.6	302.2	324.5	333.6
150	318.3	324.3	324.2	328.5	328.2	332.5	324.5	302.1	329.1	298.1	326.2	301.5	324.0	331.1
180	320.5	322.4	324.8	329.2	326.1	330.6	325.5	301.3	327.3	299.0	324.3	301.6	324.6	330.3
210	321.7	323.4	323.8	330.3	324.0	329.5	324.3	301.2	327.7	299.7	328.8	304.0	324.2	328.8
240	323.3	323.0	325.1	330.7	322.2	328.5	325.9	301.8	327.4	299.5	327.5	302.5	323.5	328.9
Average	**320**	**323**	**327**	**330**	**327**	**320**	**327**	**305**	**328**	**300**	**326**	**302**	**326**	**330**
H2SO4 conc (g/l)	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82	602.82
Sample weight (g)	400.07	400.07	400.06	400.16	400.15	400.16	400.02	400.06	400.03	400.04	400.15	400.05	400.06	400.10
Acid to pH @ t=0 (ml)	4.90	4.00	5.50	5.50	5.20	4.90	7.10	7.80	4.40	4.70	5.50	5.50	7.50	6.30
Total Leach acid (ml)	7.20	7.50	79.10	78.10	240.60	237.30	51.80	54.00	33.30	34.80	12.90	15.90	33.80	36.40
PLS volume (ml)	910.00	910.00	1035.00	1040.00	990.00	970.00	985.00	940.00	990.00	915.00	1035.00	935.00	1105.00	1030.00
PLS weight (g)	911.30	914.10	1098.70	1108.90	1066.00	1031.40	1005.40	964.40	1005.10	933.50	1048.10	944.80	1119.40	1052.20
Wash water volume (ml)	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00
Wash water weight (g)	984.00	983.90	984.10	982.60			983.30	983.30	986.70	982.10	981.40	983.40	979.70	982.40
Wash filtrate volume (ml)	1025.00	1020.00	990.00	960.00	1040.00	1070.00	1030.00	1010.00	980.00	980.00	960.00	985.00	950.00	960.00
Wash filtrate weight (g)	1021.80	1014.60	982.80	955.70	1040.00	1082.80	1027.60	1008.50	973.10	973.80	955.30	976.10	947.50	952.90
Filter paper weight (g)	3.53	3.59	3.59	3.61	3.56	3.56	3.55	3.52	3.59	3.52	3.55	3.59	3.58	3.66

P_{80}	150													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
Wet cake + filter paper (g)	483.50	481.00	433.90	440.30	574.80	654.30	525.60	517.60	516.50	516.30	507.90	495.60	481.40	465.90
Dry cake weight (g)	393.20	393.96	347.00	347.00	372.28	376.96	382.09	383.48	382.01	382.97	386.94	386.02	381.03	380.70
Feed solids (S-samples)														
T-Cu, %	0.843	0.7543	0.2676	0.2551	1.28	1.4497	0.8658	0.8569	1.33	0.28	6.8836	6.4882	1.6426	1.4669
OX-Cu, %	0.8312	0.74	0.2623	0.25144	0.8791	1.1473	0.7958	0.8357	1.3	0.17	6.7641	6.45	1.6188	1.442
% OX-Cu	99%	98%	98%	99%	69%	79%	92%	98%	98%	61%	98%	99%	99%	98%
T-Co, %	0.3046	0.2505	0.1678	0.1855	0.1194	0.1074	0.1951	0.2168	0.15	0.09	0.2322	0.1955	0.1268	0.1161
OX-Co, %	0.2375	0.1606	0.1597	0.136	0.0293	0.1041	0.1638	0.1696	0.12	0.08	0.1411	0.1387	0.114	0.0549
Fe, %	3.2842	3.2064	1.1537	1.1917	1.5472	1.5198	1.8693	1.898	2.26	1.12	1.0109	0.9325	0.6481	0.6685
Leach residue (C-samples)														
T-Cu, %	0.0821	0.01	0.0479	0.0563	0.5399	0.439	0.1306	0.1325	0.18	0.18	0.1996	0.2441	0.7397	0.0474
OX-Cu, %	0.0629	0.01	0.0457	0.0276	0.2042	0.2621	0.116	0.114	0.07	0.07	0.18	0.22	0.0905	0.0337
% OX-Cu	77%	-	95%	49%	38%	60%	89%	86%	39%	39%	90%	90%	12%	71%
T-Co, %	0.2499	0.1558	0.1698	0.1211	0.02	0.0215	0.1641	0.2069	0.09	0.1	0.2536	0.1772	0.1053	0.0781
OX-Co, %	0.1576	0.0926	0.1299	0.124	0.0184	0.0206	0.1616	0.1458	0.06	0.05	0.112	0.0853	0.0782	0.0782
Fe, %	3.3828	3.1231	0.8682	0.9537	1.4694	1.4978	2.0768	2.074	2.36	2.33	1.1395	1.164	0.6444	0.6756
PLS (PLS-samples)														
pH	1.59	1.44	1.53	1.57	1.5	1.43	1.57	1.45	1.62	1.5	1.49	1.48	1.54	1.48
Free Acid g/L	4.16	4.92	2.59	2.59	4.55	5.33	3.76	2.98	3.69	4.77	4.94	4.55	4.55	2.98
Cu (g/L)	2.9318	2.9171	0.9073	0.9121	2.7837	0.7882	2.7022	2.9684	4.8049	4.9499	23.2625	23.7576	5.8746	6.1383
Co (g/L)	0.4005	0.4843	0.1709	0.1563	0.4164	0.3528	0.1929	0.2198	0.3963	0.4031	0.367	0.4159	0.2833	0.2979
Fe (g/L)	0.0978	0.1332	0.064	0.06	0.4251	0.1424	0.0718	0.1116	0.1833	0.1999	0.0339	0.0142	0.1911	0.2014
Wash water (WW-samples)														
pH	2.07	1.95	1.99	2.00	2.06	1.88	2.25	2.12	2.08	1.99	2.03	2.26	2.75	2.39
Free Acid g/L	0.01	0.01	0.01	0.01	0.01	1.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Cu (g/L)	0.3639	0.3045	0.1145	0.1208	0.6293	2.5808	0.4652	0.5214	0.5632	0.7159	1.7911	1.6261	0.1966	0.2992
Co (g/L)	0.083	0.0808	0.0202	0.0223	0.1226	0.3821	0.0366	0.0488	0.0393	0.0486	0.0588	0.0539	0.0366	0.0342
Fe (g/L)	0.0314	0.0199	0.0103	0.0121	0.1053	0.4232	0.0206	0.049	0.0156	0.022	0.01	0.0346	0.0274	0.0392
Cu Accountability	**100 %**	**100 %**	**114 %**	**123 %**	**106 %**	**89%**	**105 %**	**112 %**	**113 %**	**529 %**	**96%**	**95%**	**145 %**	**116 %**
Leach Efficiencies														
T-Cu recovery (solids)	**90%**	**99%**	**84%**	**81%**	**61%**	**71%**	**86%**	**85%**	**87%**	**38%**	**97%**	**96%**	**57%**	**97%**
OX-Cu recovery (solids)	**93%**	**99%**	**85%**	**90%**	**78%**	**78%**	**86%**	**87%**	**95%**	**61%**	**97%**	**97%**	**95%**	**98%**

P_{80}	150													
Ore ID	RATLILLAS		RATGR		DSTRAT		SDS		SDB		RSF		RSC	
T-Co recovery (solids)	19%	39%	12%	43%	84%	81%	20%	9%	43%	-6%	-6%	13%	21%	36%
OX-Co recovery (solids)	35%	43%	29%	21%	42%	81%	6%	18%	52%	40%	23%	41%	35%	-36%
Fe recovery (solids)	-1%	4%	35%	31%	12%	7%	-6%	-5%	0%	-99%	-9%	-20%	5%	4%
Solids mass reduction, %	1.7%	1.5%	13.3%	13.3%	7.0%	5.8%	4.5%	4.1%	4.5%	4.3%	3.3%	3.5%	4.8%	4.8%
TAC (kg/t - ore))	8.74	6.11	120.75	119.18	359.01	349.39	79.48	86.09	47.65	48.59	14.92	21.59	49.64	56.64
GAC (kg/t-ore) [Cu Solids]	-3.02	-5.38	117.26	116.00	347.01	333.40	68.04	74.83	29.78	46.93	-88.35	-74.92	35.16	34.69
FAC (kg/t-ore) [Cu Solids]	0.98	-1.47	118.45	117.08	351.09	338.84	71.93	78.66	35.86	47.49	-53.24	-42.11	40.08	42.16

Table 31: Variante 212µm leach test results

P_{80}	212													
Ore ID	RATLILLAS		RATGR		DSTRAT		SDS		SDB		RSF		RSC	
Test ID	WOL - T - 24		WOL - T - 25		WOL - T - 27		WOL - T - 26		WOL - T - 22		WOL - T - 23		WOL - T - 21	
Samples	A	B	A	B	A	B	A	B	A	B	A	B	A	B
Time (minutes)	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH
0	1.490	1.450	1.400	1.484	1.434	1.398	1.480	1.470	1.520	1.440	1.481	1.503	1.410	1.510
10	1.420	1.500	1.506	1.571	1.538	1.659	1.390	1.430	1.510	1.440	1.443	1.477	1.521	1.472
20	1.470	1.460	1.468	1.503	1.480	1.496	1.490	1.510	1.500	1.500	1.491	1.501	1.509	1.471
30	1.490	1.480	1.496	1.497	1.495	1.490	1.490	1.510	1.490	1.490	1.489	1.491	1.430	1.471
60	1.500	1.500	1.503	1.500	1.493	1.485	1.510	1.490	1.490	1.490	1.497	1.499	1.492	1.498
90	1.500	1.490	1.493	1.500	1.501	1.498	1.500	1.490	1.490	1.480	1.479	1.477	1.495	1.485
120	1.500	1.490	1.503	1.498	1.498	1.505	1.500	1.500	1.490	1.490	1.489	1.472	1.480	1.498
150	1.500	1.460	1.500	1.497	1.488	1.491	1.500	1.500	1.480	1.490	1.474	1.480	1.486	1.495
180	1.500	1.460	1.494	1.497	1.489	1.484	1.470	1.520	1.500	1.490	1.493	1.491	1.488	1.501
210	1.500	1.460	1.494	1.495	1.490	1.496	1.490	1.490	1.480	1.480	1.505	1.492	1.500	1.495
240	1.500	1.450	1.497	1.500	1.498	1.492	1.500	1.500	1.500	1.500	1.483	1.496	1.502	1.502
Average	1.488	1.473	1.487	1.504	1.491	1.499	1.484	1.492	1.495	1.481	1.484	1.489	1.483	1.491
Time (minutes)	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP
0	324.4	324.7	321.9	320.2	325.2	329.2	325.8	324	328.2	327.2	322.6	321.9	325.8	322.6
10	338.5	332.4	326.9	326.1	326.5	323.3	340.3	335.8	333.9	334.4	328.7	329.9	320.9	329.8
20	337.8	337.7	328.5	332.0	330.0	334.5	335.4	333.1	334.3	331.7	325.1	328.6	321.0	329.4

P_{80}	212													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
30	336.0	335.7	326.0	331.8	328.1	333.8	335.1	333.5	333.6	331.6	324.1	328.4	325.0	328.6
60	332.3	331.8	322.9	328.6	324.6	330.2	332.5	332.5	331.1	329.7	321.2	325.5	328.3	326.5
90	330.3	330.0	322.1	326.7	321.9	336.9	330.8	330.1	332.7	330.7	325.1	331.0	324.8	327.2
120	332.6	328.8	324.3	328.9	322.4	326.6	330.7	329.0	331.3	329.0	324.2	328.6	323.1	324.6
150	331.5	332.3	322.7	328.6	324.5	325.1	331.5	328.6	333.6	331.6	324.2	328.3	324.8	327.4
180	332.8	332.3	325.3	326.8	321.6	326.1	331.5	328.2	332.6	330.5	324.6	328.1	322.3	325.6
210	331.7	332.5	323.6	325.7	323.7	326.8	332.2	329.4	332.3	330.4	324.9	327.3	323.8	327.1
240	330.5	331.7	321.9	328.1	324.3	326.8	330.5	329.6	335.4	332.7	323.2	328.5	321.7	325.1
Average	**332.6**	**331.8**	**324.2**	**327.6**	**324.8**	**329.0**	**332.4**	**330.3**	**332.6**	**330.9**	**324.4**	**327.8**	**323.8**	**326.7**
H2SO4 conc (g/l)	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79	590.79
Sample weight (g)	400.33	400.70	400.40	400.39	400.37	400.35	400.25	400.18	400.01	400.04	400.17	400.45	400.11	400.07
Acid to pH @ t=0 (ml)	5.30	4.20	5.40	5.20	5.10	6.40	5.10	4.20	4.80	5.30	4.30	5.10	4.90	4.80
Total Leach acid (ml)	12.90	11.10	81.50	84.20	111.90	111.80	63.00	60.40	28.60	26.10	18.50	20.20	27.30	28.40
PLS volume (ml)	880.00	950.00	1025.00	975.00	990.00	1050.00	950.00	980.00	970.00	970.00	960.00	940.00	965.00	945.00
PLS weight (g)	874.20	942.20	1077.20	1019.20	1080.80	1143.70	965.20	997.40	976.40	978.50	954.90	942.80	972.30	953.30
Wash water volume (ml)	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00	1000.00
Wash water weight (g)	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25	998.25
Wash filtrate volume (ml)	945.00	965.00	1030.00	1065.00	973.00	1010.00	970.00	980.00	980.00	980.00	990.00	975.00	980.00	1000.00
Wash filtrate weight (g)	919.60	949.40	1025.80	1054.10	963.60	1002.30	950.10	967.40	965.50	969.20	976.90	955.50	968.40	986.30
Filter paper weight (g)	3.67	3.69	3.67	3.63	3.36	3.63	3.64	3.65	3.61	3.60	3.66	3.68	3.66	3.61
Wet cake + filter paper (g)	492.60	488.60	432.40	428.70	389.20	386.90	510.20	497.80	466.40	480.70	491.50	468.70	449.90	462.20
Dry cake weight (g)	392.60	392.60	346.70	344.90	318.54	318.07	378.96	379.25	386.60	386.30	389.60	390.20	384.20	383.80
Feed solids (S-samples)														
T-Cu, %	0.2599	0.2694	6.6592	7.5791	10.6689	10.3302	0.9104	0.8393	1.2078	1.1844	0.9355	0.8916	1.5522	1.3662
OX-Cu, %	0.2591	0.2568	6.6081	7.4937	10.01	10.23	0.7787	0.7858	1.1399	1.1795	0.933	0.8903	1.2313	1.2832
% OX-Cu	100%	95%	99%	99%	94%	99%	86%	94%	94%	100%	100%	100%	79%	94%
T-Co, %	0.1325	0.1064	0.1872	0.1953	0.306	0.3186	0.1592	0.1763	0.1091	0.1119	0.1341	0.1128	0.1185	0.1246
OX-Co, %	0.0429	0.0448	0.1304	0.1148	0.2881	0.2641	0.1115	0.1374	0.074	0.0604	0.0623	0.0448	0.0908	0.0786
Fe, %	NA	NA	NA	NA	0.4499	0.4837	2.7753	1.9998	NA	NA	NA	NA	0.6244	0.5667
Leach residue (C-samples)														
T-Cu, %	0.0341	0.0395	0.1756	0.2811	0.2414	0.1919	0.126	0.0936	0.0433	0.0415		0.1037	0.3019	0.2836

P_{80}	212													
Ore ID	**RATLILLAS**		**RATGR**		**DSTRAT**		**SDS**		**SDB**		**RSF**		**RSC**	
OX-Cu, %	0.0234	0.0368	0.112	0.1124	0.2033	0.18	0.1167	0.08	0.0132	0.022		0.0556	0.0573	0.0504
% OX-Cu	69%	93%	64%	40%	84%	94%	93%	85%	30%	53%		54%	19%	18%
T-Co, %	0.0782	0.0887	0.1239	0.121	0.1962	0.1911	0.1276	0.0982	0.0651	0.0534		0.0582	0.0075	0.0694
OX-Co, %	0.0026	0.0351	0.0526	0.0468	0.20 06	0.18	0.1294	0.1115	0.05	0.02		0.01	0.0012	0.0342
Fe, %	NA	NA	NA	NA	0.5661	0.5626	2.0112	2.0191	NA	NA		NA	0.5492	0.519
PLS (PLS-samples)														
pH	1.54	1.63	1.61	1.51	1.53	NA	1.52	1.7	1.46	1.6	1.59	1.5	1.7	1.62
Free Acid g/L	3.8	2.61	3.4	4.59	4.16	NA	4.94	2.98	4.99	3.4	3.4	4.2	3.41	4.2
Cu (g/L)	0.9882	0.9166	23.8716	24.5101	38.3006	36.3342	2.9212	2.9001	4.6689	4.501	3.5065	3.6331	5.4048	5.2715
Co (g/L)	0.2883	0.2677	0.447	0.4646	0.7115	0.6993	0.2699	0.3581	0.3882	0.3823	0.3588	0.3823	0.4037	0.4095
Fe (g/L)	0.0067	0.0063	0.0052	0.0082	0.11	0.1051	0.1486	0.1076	0.0779	0.0583	0.074	0.0858	NA	NA
Wash water (WW-samples)														
pH	NA	NA	NA	NA	NA	2.22	NA	NA	NA	NA	NA	NA	NA	NA
Free Acid g/L	NA	NA	NA	NA	NA	0.01	NA	NA	NA	NA	NA	NA	NA	NA
Cu (g/L)	0.1102	0.068	2.9607	3.5487	2.6918	2.7861	0.1844	0.2576	0.2864	0.2809	0.2854	0.3047	0.3952	0.4413
Co (g/L)	0.059	0.0649	0.0767	0.1178	0.0983	0.0199	0.1032	0.0003	0.059	0.0649	0.1031	0.1207	0.0949	0.103
Fe (g/L)	0.3403	0.3167	0.5693	0.8699	0.0176	0.0117	0.0078	0.0137	0.4248	0.2922	0.528	0.8572	0.009	0.01
Cu Accountability	**106 %**	**101 %**	**105 %**	**94%**	**97%**	**101 %**	**94%**	**103 %**	**103 %**	**101 %**	**97%**	**115 %**	**109 %**	**119 %**
Leach Efficiencies														
T-Cu recovery (solids)	**87%**	**86%**	**98%**	**97%**	**98%**	**99%**	**87%**	**89%**	**97%**	**97%**	**100 %**	**89%**	**81%**	**80%**
OX-Cu recovery (solids)	**91%**	**86%**	**99%**	**99%**	**98%**	**99%**	**86%**	**90%**	**99%**	**98%**	**100 %**	**94%**	**96%**	**96%**
T-Co recovery (solids)	**42%**	**18%**	**43%**	**47%**	**49%**	**52%**	**24%**	**47%**	**42%**	**54%**	**100 %**	**50%**	**94%**	**47%**
OX-Co recovery (solids)	**94%**	**23%**	**65%**	**65%**	**45%**	**46%**	**-10%**	**23%**	**35%**	**68%**	**100 %**	**78%**	**99%**	**58%**
Fe recovery (solids)	**NR**	**NR**	**NR**	**NR**	**0%**	**8%**	**31%**	**4%**	**NR**	**NR**	**NR**	**NR**	**16%**	**12%**
Solids mass reduction, %	**1.9%**	**2.0%**	**13.4 %**	**13.9 %**	**20.4 %**	**20.6 %**	**5.3%**	**5.2%**	**3.4%**	**3.4%**	**2.6%**	**2.6%**	**4.0%**	**4.1%**
TAC (kg/t - ore))	**19.04**	**16.37**	**120.25**	**124.24**	**165.12**	**164.98**	**92.99**	**89.17**	**42.24**	**38.55**	**27.31**	**29.80**	**40.31**	**41.94**
GAC (kg/t-ore) [Cu Solids]	**15.54**	**12.81**	**19.82**	**11.00**	**3.42**	**7.89**	**80.78**	**77.58**	**24.24**	**20.88**	**12.87**	**17.60**	**20.83**	**25.05**
FAC (kg/t-ore) [Cu Solids]	**16.73**	**14.02**	**53.97**	**49.50**	**58.40**	**61.30**	**84.93**	**81.52**	**30.36**	**26.89**	**17.78**	**21.75**	**27.45**	**30.79**

Key Variable factor tests results (pH, Temperature, Percentage solids) done on core sample blend

Test Description	Base line		Acid only		Low pH targeted		High pH targeted		Low %solids		High %solids		Med temperature		High temperature	
Test ID	WOL-B-01		WOL-B-02		WOL-B-03		WOL-B-04		WOL-B-07		WOL-B-08		WOL-B-09		WOL-B-10	
Samples	A	B	A	B	A	B	A	B	A	B	A	B	A	B	A	B
Time (minutes)	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH	pH
0	1.430	1.445	1.494	1.510	0.960	0.930	1.992	2.050	1.520	1.497	1.510	1.450	1.410	1.459	1.502	1.511
10	1.430	1.484	1.476	1.470	1.000	0.991	2.030	2.020	1.220	1.151	1.249	1.090	1.510	1.509	1.550	1.520
20	1.500	1.471	1.411	1.400	0.990	1.002	1.955	2.020	1.450	1.404	1.484	1.500	1.430	1.458	1.487	1.530
30	1.510	1.509	1.443	1.480	1.000	1.001	1.869	1.810	1.520	1.502	1.492	1.480	1.490	1.504	1.500	1.480
60	1.510	1.503	1.504	1.500	1.010	0.998	1.995	1.980	1.500	1.501	1.453	1.330	1.370	1.467	1.491	1.490
90	1.500	1.514	1.497	1.510	1.000	1.001	1.893	1.980	1.500	1.521	1.465	1.520	1.490	1.494	1.462	1.510
120	1.500	1.504	1.508	1.500	1.010	0.944	2.057	1.920	1.490	1.500	1.538	1.500	1.490	1.504	1.463	1.500
150	1.570	1.552	1.497	1.500	0.990	0.994	2.042	2.020	1.500	1.500	1.480	1.450	1.480	1.482	1.427	1.490
180	1.700	1.585	1.505	1.500	1.000	0.996	2.006	2.000	1.500	1.500	1.499	1.490	1.450	1.483	1.483	1.490
210	1.490	1.493	1.507	1.490	1.000	0.994	1.883	2.020	1.500	1.504	1.501	1.480	1.490	1.498	1.495	1.495
240	1.520	1.497	1.506	1.510	1.000	1.004	2.034	2.010	1.500	1.506	1.513	1.500	1.500	1.501	1.401	1.470
Average	1.5	1.5	1.5	1.5	1.0	1.0	2.0	2.0	1.5	1.5	1.5	1.4	1.5	1.5	1.5	1.5
Time (minutes)	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP	ORP
150	331	366	344.50		323	322	384	372	371	372	378	374	377	385	367	360
180	330	380	352.31		333	334	354	361	374	373	376	372	374	386	365	364
210	355	375	366.50		347	347	372	378	376	381	372	371	371	373	371	377
240	375	374	371.00	539	357	364	358	357	371	372	371	370	373	375	365	364
Average	348	374	359	539	340	342	387	395	373	375	374	372	374	381	373	369
Target pH	1.5	1.5	1.5	1.5	1.0	1.0	2.0	2.0	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Target Eh	375	375	375	375	375	375	375	375	375	375	375	375	375	375	375	375
Target % solids	30%	30%	30%	30%	30%	30%	30%	30%	20%	20%	40%	40%	30%	30%	30%	30%
Target temperature (C)	30	30	30	30	30	30	30	30	30	30	30	30	45	45	60	60
H2SO4 conc (g/L)	196.00	196.00	196.00	196.00	196.00	196.00	196.00	196.00	196.00	196.00	196.00	196.00	196.00	196.00	196.00	196.00
SO2 conc (g/L)	90.00	90.00	0.00	0.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00

Test Description	Base line		Acid only		Low pH targeted		High pH targeted		Low %solids		High %solids		Med temperature		High temperature	
Test ID	WOL-B-01		WOL-B-02		WOL-B-03		WOL-B-04		WOL-B-07		WOL-B-08		WOL-B-09		WOL-B-10	
Samples	A	B	A	B	A	B	A	B	A	B	A	B	A	B	A	B
SO2 soln density, g/L	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000
Sample weight (g)	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00
Initial solution (g)	933.33	933.33	933.33	933.33	933.33	933.33	933.33	933.33	1600.00	1600.00	600.00	600.00	933.33	933.33	933.33	933.33
%solids	30%	30%	30%	30%	30%	30%	30%	30%	20%	20%	40%	40%	30%	30%	30%	30%
Sample weight (g)	400.15	400.86	400.81	401.71	400.28	400.28	400.88	400.59	400.72	401.58	400.45	401.60	401.55	401.97	401.03	401.68
H2SO4 added to achieve pH @ t=0 (mL)	15.40	17.90	14.60	14.20	52.50	68.70	5.30	4.90	26.10	26.00	9.10	11.80	20.40	20.10	16.90	16.10
Total H2SO4 added (mL)	284.10	293.20	289.90	298.30	330.80	323.10	247.60	252.40	298.60	298.30	286.10	280.40	314.90	313.30	328.30	325.20
Total H_2SO_3 added (g)	80.74	120.33			190.81	332.98	30.54	26.18	42.70	36.90	85.50	55.53	45.95	63.49	28.68	28.04
PLS volume (mL)	1280.00	1320.00	1210.00	1150.00	1430.00	1550.00	1280.00	1210.00	1810.00	1730.00	960.00	940.00	1170.00	1190.00	1115.00	1070.00
PLS weight (g)	1330.20	1378.80	1251.10	1193.00	1483.20	1600.30	1313.00	1251.70	1841.70	1763.40	1003.00	983.00	1216.50	1242.40	1160.50	1117.50
Wash water volume (mL)	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00	1040.00
Wash filtrate volume (mL)	990.00	990.00	940.00	1000.00	1000.00	980.00	990.00	1010.00	970.00	950.00	1000.00	980.00	1000.00	970.00	1020.00	980.00
Wash filtrate weight (g)	976.70	978.60	932.90	990.10	988.70	967.20	970.70	1004.70	956.70	940.60	982.00	970.20	985.90	953.10	1005.90	977.30
Filter paper weight (g)	3.66	3.61	3.59	3.55	3.57	3.61	3.61	3.61	3.61	3.62	3.61	3.61	3.61	3.61	3.61	3.61
Wet cake + filter paper (g)	454.60	455.40	465.60	482.10	476.30	482.60	466.20	453.30	472.10	455.40	441.50	467.00	466.30	467.60	448.60	459.10
Dry cake weight (g)	359.10	359.20	359.80	361.90	357.00	358.20	362.70	361.60	360.20	358.42	357.26	361.41	359.23	358.71	359.38	360.97
T-Cu, %	4.18	4.20	4.18	4.20	4.18	4.20	4.18	4.20	4.18	4.20	4.18	4.20	4.18	4.20	4.18	4.20
OX-Cu, %	3.96	4.11	3.96	4.11	3.96	4.11	3.96	4.11	3.96	4.11	3.96	4.11	3.96	4.11	3.96	4.11
% OX-Cu	95%	98%	95%	98%	95%	98%	95%	98%	95%	98%	95%	98%	95%	98%	95%	98%
T-Co, %	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39
OX-Co, %	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36
Fe, %	0.59	0.62	0.59	0.62	0.59	0.62	0.59	0.62	0.59	0.62	0.59	0.62	0.59	0.62	0.59	0.62
Leach residue (C-samples)																

Test Description	Base line		Acid only		Low pH targeted		High pH targeted		Low %solids		High %solids		Med temperature		High temperature	
Test ID	WOL-B-01		WOL-B-02		WOL-B-03		WOL-B-04		WOL-B-07		WOL-B-08		WOL-B-09		WOL-B-10	
Samples	A	B	A	B	A	B	A	B	A	B	A	B	A	B	A	B
T-Cu, %	0.20	0.24	0.22	0.28	0.19	0.24	0.31	0.34	0.62	0.28	0.23	0.27	0.22	0.30	0.21	0.21
OX-Cu, %	0.03	0.01	0.14	0.03	0.11	0.12	0.21	0.24	0.11	0.11	0.14	0.13	0.08	0.08	0.07	0.07
% OX-Cu	15%	4%	64%	11%	59%	49%	70%	70%	17%	40%	63%	46%	38%	28%	34%	31%
T-Co, %	0.08	0.12	0.19	0.15	0.07	0.08	0.16	0.18	0.06	0.07	0.08	0.07	0.05	0.06	0.06	0.04
OX-Co, %	0.00	0.00	0.12	0.00	0.02	0.03	0.12	0.10	0.03	0.03	0.03	0.01	0.05	0.05	0.04	0.05
Fe, %	0.66	0.51	0.67	0.58	0.58	0.49	0.63	0.61	0.54	0.64	0.55	0.60	0.59	0.68	0.61	0.57
PLS (PLS-samples)																
pH	1.63	1.56	1.62	1.60	1.11	1.08	2.38	2.37	1.61	1.61	1.66	1.85	1.48	1.48	1.41	1.45
Free Acid g/L	1.98	1.98	1.98	1.98	2.00	2.08	0.00	0.00	3.56	3.56	3.56	2.52	6.52	6.48	7.52	7.50
Cu (g/L)	12.35	11.75	12.86	13.71	11.13	9.76	13.73	11.75	8.24	9.07	15.53	16.48	15.54	15.41	14.17	14.76
Co (g/L)	1.10	1.08	0.88	0.94	1.03	0.93	1.04	0.94	0.81	0.82	1.32	2.66	1.74	1.22	1.04	1.19
Fe (g/L)	0.19	0.17	0.13	0.16	0.22	0.20	0.12	0.08	0.06	0.06	0.11	0.18	0.19	0.26	0.15	0.18
Wash water (WW-samples)																
pH																
Free Acid g/L	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Cu (g/L)	0.93	0.85	0.81	1.48	0.58	1.08	0.91	1.40	0.82	0.94	1.33	1.34	1.21	1.81	1.22	1.18
Co (g/L)	0.07	0.07	0.05	0.08	0.05	0.04	0.09	0.11	0.08	0.07	0.11	0.33	0.12	0.18	0.15	0.14
Fe (g/L)	0.02	0.01	0.02	0.03	0.03	0.02	0.01	0.03	0.01	0.01	0.01	0.01	0.00	0.01	0.01	0.01
H2SO4 Acid consumed, g	56.168	58.362	57.286	58.973	72.267	73.569	49.568	50.431	57.198	57.404	54.442	54.902	58.090	57.635	59.274	58.870
SO2 consumed, g	7.267	10.830	0.000	0.000	17.173	29.968	2.749	2.356	3.843	3.321	7.695	4.998	4.136	5.714	2.581	2.524
T-Cu in solids at t0, g	16.726	16.836	16.754	16.872	16.732	16.812	16.757	16.825	16.750	16.866	16.739	16.867	16.785	16.883	16.763	16.871
OX-Cu in solids at t0, g	15.846	16.475	15.872	16.510	15.851	16.452	15.875	16.464	15.869	16.505	15.858	16.506	15.901	16.521	15.881	16.509
T-Co in solids at t0, g	1.557	1.547	1.560	1.550	1.557	1.545	1.560	1.546	1.559	1.550	1.558	1.550	1.562	1.551	1.560	1.550
OX-Co in solids at t0, g	1.423	1.459	1.425	1.462	1.423	1.457	1.426	1.458	1.425	1.461	1.424	1.461	1.428	1.463	1.426	1.462
Fe in solids at t0, g	2.354	2.469	2.358	2.474	2.355	2.465	2.359	2.467	2.358	2.473	2.356	2.473	2.363	2.476	2.360	2.474
T-Cu in residue, g	0.710	0.871	0.792	1.021	0.689	0.869	1.115	1.225	2.222	0.990	0.813	0.989	0.774	1.058	0.756	0.765
OX-Cu in residue, g	0.107	0.037	0.508	0.108	0.405	0.428	0.778	0.853	0.386	0.398	0.513	0.455	0.296	0.294	0.255	0.239
T-Co in residue, g	0.303	0.440	0.693	0.544	0.252	0.298	0.598	0.645	0.219	0.249	0.273	0.260	0.182	0.218	0.210	0.152

Test Description	Base line		Acid only		Low pH targeted		High pH targeted		Low %solids		High %solids		Med temperature		High temperature	
Test ID	WOL-B-01		WOL-B-02		WOL-B-03		WOL-B-04		WOL-B-07		WOL-B-08		WOL-B-09		WOL-B-10	
Samples	A	B	A	B	A	B	A	B	A	B	A	B	A	B	A	B
OX-Co in residue, g	0.000	0.000	0.444	0.004	0.067	0.120	0.436	0.375	0.107	0.092	0.106	0.029	0.163	0.184	0.128	0.196
Fe in residue, g	2.376	1.828	2.394	2.105	2.073	1.762	2.290	2.213	1.953	2.307	1.953	2.156	2.130	2.450	2.176	2.050
Cu in soln (PLS+WW), g	16.730	16.349	16.320	17.245	16.495	16.181	18.476	15.628	15.716	16.584	16.233	16.803	19.398	20.098	17.035	16.956
Co in soln (PLS+WW), g	1.479	1.497	1.108	1.161	1.528	1.482	1.424	1.253	1.537	1.489	1.379	2.819	2.148	1.622	1.310	1.414
Fe in soln (PLS+WW), g	0.258	0.227	0.183	0.213	0.335	0.330	0.157	0.126	0.119	0.105	0.121	0.185	0.225	0.317	0.181	0.199
Cu Acountability	104%	102%	102%	108%	103%	101%	117%	100%	107%	104%	102%	105%	120%	125%	106%	105%
T-Cu recovery (solids)	96%	95%	95%	94%	96%	95%	93%	93%	87%	94%	95%	94%	95%	94%	95%	95%
OX-Cu recovery (solids)	99%	100%	97%	99%	97%	97%	95%	95%	98%	98%	97%	97%	98%	98%	98%	99%
T-Co recovery (solids)	81%	72%	56%	65%	84%	81%	62%	58%	86%	84%	83%	83%	88%	86%	87%	90%
OX-Co recovery (solids)	100%	100%	69%	100%	95%	92%	69%	74%	92%	94%	93%	98%	89%	87%	91%	87%
Fe recovery (solids)	-1%	26%	-2%	15%	12%	29%	3%	10%	17%	7%	17%	13%	10%	1%	8%	17%
Solids mass reduction, %	10.3%	10.4%	10.2%	9.9%	10.8%	10.5%	9.5%	9.7%	10.1%	10.7%	10.8%	10.0%	10.5%	10.8%	10.4%	10.1%
TAC (kg/t - ore))	140.37	145.59	142.93	146.80	180.54	183.79	123.65	125.89	142.74	142.95	135.95	136.71	144.67	143.38	147.81	146.56
GAC (kg/t-ore) [O-Cu Solids]	79.66	82.30	83.76	83.78	120.98	122.01	65.52	65.74	83.11	81.04	76.81	75.02	84.68	81.08	87.67	84.04
FAC (kg/t-ore) [O-Cu Solids]	100.30	103.82	103.88	105.21	141.23	143.01	85.29	86.19	103.38	102.09	96.92	96.00	105.08	102.26	108.11	105.30

Test results from drilling campaign

Table 32: All each test results from tests performed on drilling campaign samples to obtain a better understanding of the behaviour of the leach characteristics of the deposit

Fragment	Lithology	Head Grade TCu%	Head Grade TCo%	Leach TCu Recovery	Leach CuOx Recovery	Leach TCo Recovery	TAC	GAC	FAC
VARIANTE	RSF	1.90	0.09	97%	98%	48%	31	4	14
FNSR	BOMZ	0.67	0.18	72%	85%	13%	21	13	16
FNSR	DSTRAT	12.85	0.66	95%	95%	13%	188	6	68
FNSR	RATGRIS	15.08	0.07	98%	99%	45%	236	15	90
FNSR	RSC	1.80	0.53	93%	96%	21%	96	73	81
FNSR	RSF	3.48	2.26	87%	91%	9%	51	7	22
FNSR	SDB	8.94	0.82	96%	97%	14%	135	8	52
FNSR	BOMZ	8.89	4.76	88%	88%	3%	148	36	74
VIRGULE	BOMZ	3.73	1.18	66%	74%	4%	54	19	31
VIRGULE	BRECHE	7.16	0.04	96%	97%	29%	124	28	61
FNSR	DSTRAT	6.70	0.15	98%	98%	28%	114	23	54
VIRGULE	DSTRAT	11.06	0.80	97%	97%	12%	200	44	97
FNSR	RATGRIS	8.65	0.05	97%	97%	42%	145	31	70
VIRGULE	RATGRIS	11.21	0.06	97%	98%	46%	181	41	88
FNSR	RSC	4.01	0.15	95%	98%	55%	58	10	27
VIRGULE	RSC	2.06	0.58	95%	96%	16%	34	9	18
FNSR	RSF	7.45	0.06	94%	96%	21%	118	21	54
VIRGULE	RSF	6.97	0.77	97%	97%	3%	122	24	57
FNSR	SDB	10.59	0.12	90%	97%	87%	148	17	61
VIRGULE	SDB	7.02	0.81	93%	95%	8%	119	22	55
FNSR	SDS	1.77	0.16	91%	94%	30%	270	243	252
FNSR	BOMZ	7.55	0.49	98%	98%	11%	124	21	56
FNSR	BR HET	1.80	0.11	94%	95%	17%	142	116	125
FNSR	DSTRAT	8.66	0.18	96%	97%	26%	185	62	104
FNSR	RATGRIS	8.48	0.11	96%	96%	37%	148	32	71
FNSR	RSC	2.61	0.40	95%	96%	31%	40	6	18
FNSR	RSF	5.22	0.64	95%	97%	39%	91	20	45
FNSR	SDB	8.89	0.27	98%	98%	35%	150	23	66
FNSR	SDS	2.56	0.25	100%	100%	101%	55	18	31
FNSR	BOMZ	2.02	2.89	72%	78%	15%	10	3	5
FNSR	DSTRAT	6.74	0.03	92%	97%	67%	80	10	34
FNSR	RATGRIS	7.46	0.04	53%	96%	36%	452	424	434
FNSR	RATL	4.49	0.03	7%	76%	21%	347	344	345
FNSR	RSC	6.38	0.05	78%	86%	28%	394	391	392
FNSR	RSF	3.29	0.26	-35%	94%	83%	304	288	293
FNSR	SDB	1.18	0.81	24%	40%	36%	837	837	837
FNSR	BOMZ	1.03	0.80	100%	100%	100%	746	734	739
FNSR	DSTRAT	7.20	0.03	63%	94%	41%	438	401	414
FNSR	RATGRIS	6.13	0.02	60%	92%	42%	315	287	296
FNSR	RSC	3.93	0.21	91%	97%	71%	116	89	98
FNSR	RSF	6.23	0.01	86%	98%	70%	73	20	38
FNSR	SDB	7.07	0.30	79%	98%	73%	103	58	73

Fragment	Lithology	Head Grade TCu%	Head Grade TCo%	Leach TCu Recovery	Leach CuOx Recovery	Leach TCo Recovery	TAC	GAC	FAC
FNSR	SDS	1.40	0.16	31%	79%	42%	460	459	459
OLIVEIRA	BOMZ	1.23	0.36	71%	83%	6%	288	273	278
OLIVEIRA	SDB	1.41	0.47	84%	87%	12%	333	315	321
OLIVEIRA	BRECHE	0.38	0.10	89%	95%	77%	460	455	457
OLIVEIRA	DSTRAT	2.70	0.34	43%	92%	59%	582	578	579
OLIVEIRA	RATGRIS	3.36	0.47	36%	90%	69%	340	330	334
OLIVEIRA	RSF	6.37	0.64	91%	96%	37%	271	217	236
OLIVEIRA	SDB	1.68	0.11	94%	96%	52%	265	240	248
FNSR	BRECHE	0.62	0.02	26%	61%	17%	394	393	393
FNSR	BR.RAT	3.13	0.01	14%	75%	10%	388	385	386
FNSR	DSTRAT	3.51	0.05	37%	88%	65%	500	494	496
FNSR	RATGRIS	5.32	0.04	8%	68%	17%	291	290	290
FNSR	RSC	1.68	0.21	60%	92%	90%	334	329	330
FNSR	RSF	4.20	0.03	31%	83%	20%	315	311	312
FNSR	SDB	9.15	0.01	3%	40%	-67%	217	216	216
OLIVEIRA	BRECHE	0.03	0.01	15%	65%	22%	448	447	448
OLIVEIRA	BOMZ	0.94	0.06	84%	90%	86%	754	742	746
OLIVEIRA	DSTRAT	8.42	0.78	96%	96%	14%	138	18	59
OLIVEIRA	RATGRIS	7.10	0.21	97%	98%	31%	114	10	45
OLIVEIRA	RSC	3.76	0.18	93%	96%	68%	52	7	22
OLIVEIRA	RSF	9.19	0.10	97%	98%	92%	141	11	55
OLIVEIRA	SDB	7.34	0.59	92%	97%	50%	172	76	109
OLIVEIRA	DSTRAT	2.65	0.09	80%	95%	85%	532	504	513
OLIVEIRA	RATGRIS	2.40	0.35	91%	94%	44%	328	294	306
OLIVEIRA	RSC	0.76	0.39	90%	94%	47%	432	422	425
OLIVEIRA	RSF	2.89	0.27	93%	95%	38%	223	187	199
OLIVEIRA	SDB	3.59	0.64	90%	91%	12%	120	73	89
OLIVEIRA	BOMZ	0.58	0.04	79%	89%	48%	661	653	19
OLIVEIRA	BOMZ	0.58	0.04	79%	89%	48%	661	653	656
OLIVEIRA	DSTRAT	5.35	0.21	45%	93%	62%	350	337	39
OLIVEIRA	DSTRAT	5.35	0.21	45%	93%	62%	350	337	341
OLIVEIRA	RATGRIS	2.97	0.40	55%	93%	40%	229	219	222
OLIVEIRA	RATL	0.05	0.06	100%	100%	100%	401	400	26
OLIVEIRA	RATL	0.05	0.06	46%	76%	46%	399	398	399
OLIVEIRA	RSC	0.44	0.42	82%	93%	45%	309	306	307
OLIVEIRA	RSF	4.40	0.57	57%	93%	31%	374	363	44
OLIVEIRA	RSF	4.40	0.57	57%	93%	31%	374	363	367
OLIVEIRA	SDB	3.03	1.59	87%	91%	16%	235	196	210
OLIVEIRA	SDS	1.70	0.12	87%	93%	26%	36	12	26
OLIVEIRA	SDS	1.70	0.12	87%	93%	26%	36	12	20
OLIVEIRA	DSTRAT	2.85	0.10	57%	92%	60%	499	486	491
OLIVEIRA	RATGRIS	1.25	0.52	57%	84%	53%	377	370	373
OLIVEIRA	RATL	0.09	0.11	63%	86%	48%	430	430	430
OLIVEIRA	RSC	0.39	0.36	87%	91%	63%	341	337	339
OLIVEIRA	RSF	4.12	0.24	88%	97%	84%	309	271	284
OLIVEIRA	SDB	3.53	0.65	87%	95%	46%	354	310	326

Fragment	Lithology	Head Grade TCu%	Head Grade TCo%	Leach TCu Recovery	Leach CuOx Recovery	Leach TCo Recovery	TAC	GAC	FAC
OLIVEIRA	SDS	0.58	0.16	89%	95%	67%	512	504	507
OLIVEIRA	BOMZ	0.60	0.05	65%	84%	63%	621	616	618
OLIVEIRA	DSTRAT	6.09	0.16	97%	98%	44%	107	15	46
OLIVEIRA	RSC	5.58	0.72	96%	97%	50%	81	10	34
OLIVEIRA	RSF	4.78	0.10	96%	99%	84%	75	7	30
OLIVEIRA	SDB	5.19	0.76	93%	95%	33%	264	190	216
OLIVEIRA	SDS	1.11	0.01	67%	92%	49%	53	45	48
OLIVEIRA	BOMZ	1.10	0.26	75%	81%	9%	40	26	31
OLIVEIRA	DSTRAT	2.94	0.86	95%	95%	10%	49	10	23
OLIVEIRA	RSC	0.52	0.71	80%	82%	10%	13	7	9
OLIVEIRA	RSF	2.20	0.18	94%	96%	62%	42	11	21
OLIVEIRA	SDB	3.29	1.24	92%	93%	14%	58	11	27
OLIVEIRA	DSTRAT	4.52	0.74	96%	96%	19%	74	11	33
OLIVEIRA	RSC	1.55	0.76	92%	93%	33%	30	9	16
OLIVEIRA	RSF	5.65	0.57	95%	97%	62%	93	13	41
OLIVEIRA	SDB	4.82	1.77	92%	93%	14%	144	75	99
OLIVEIRA	BOMZ	3.88	0.25	90%	91%	21%	215	163	181
OLIVEIRA	BRECHE	1.29	0.27	92%	95%	31%	25	7	13
OLIVEIRA	DSTRAT	5.01	0.58	97%	98%	30%	82	11	35
OLIVEIRA	RATGris	5.65	0.54	97%	97%	44%	96	12	41
OLIVEIRA	RSC	1.86	0.38	97%	98%	57%	40	14	23
OLIVEIRA	RSF	2.94	0.34	94%	97%	68%	53	11	25
OLIVEIRA	SDB	5.92	1.44	95%	96%	24%	149	62	92
OLIVEIRA	BOMZ	0.81	0.28	71%	76%	18%	14	5	8
OLIVEIRA	DSTRAT	4.86	1.36	93%	94%	19%	69	8	29
OLIVEIRA	RATGRIS	4.69	0.46	96%	97%	24%	73	12	33
OLIVEIRA	SDB	1.67	1.42	85%	87%	13%	27	4	12
OLIVEIRA	BRECHE	0.87	0.15	89%	90%	18%	19	6	11
OLIVEIRA	SDB	5.18	1.00	93%	94%	25%	90	14	40
VIRGULE	BRECHE	1.60	0.40	90%	93%	12%	28	6	14
VIRGULE	DSTRAT	5.74	0.09	97%	98%	56%	82	5	31
VIRGULE	RATGRIS	3.50	0.21	97%	98%	26%	56	7	24
VIRGULE	RSC	2.22	0.49	96%	97%	23%	41	10	21
VIRGULE	RSF	6.75	0.99	97%	97%	9%	102	8	41
VIRGULE	SDB	10.70	0.33	97%	98%	61%	163	10	11
OLIVEIRA	RSF	3.80	0.76	94%	95%	28%	67	13	32
OLIVEIRA	SDB	6.63	1.11	94%	94%	15%	201	106	138
VIRGULE	BOMZ	5.66	1.00	80%	82%	12%	82	11	35
VIRGULE	DSTRAT	4.41	0.14	96%	97%	16%	77	14	35
VIRGULE	RSC	2.43	0.13	96%	98%	72%	37	6	17
VIRGULE	RSF	10.06	0.04	96%	99%	48%	151	5	55
VIRGULE	SDB	7.90	0.77	96%	97%	20%	119	6	45
VIRGULE	SDS	3.56	0.15	90%	90%	20%	46	4	18
VARIANTE	BOMZ	0.39	0.03	43%	79%	51%	9	6	7
VARIANTE	DSTRAT	0.56	0.04	75%	84%	48%	11	6	8
VARIANTE	RATGRIS	0.52	0.05	76%	88%	24%	11	5	7

Fragment	Lithology	Head Grade TCu%	Head Grade TCo%	Leach TCu Recovery	Leach CuOx Recovery	Leach TCo Recovery	TAC	GAC	FAC
VARIANTE	RATL	0.34	0.06	85%	89%	24%	10	5	7
VARIANTE	RSC	1.09	0.09	96%	98%	22%	18	5	10
VARIANTE	RSF	0.38	0.05	94%	96%	46%	10	4	6
VARIANTE	SDB	0.31	0.04	49%	72%	34%	8	6	6
VARIANTE	SDS	0.84	0.04	70%	89%	23%	13	4	7
VARIANTE	BOMZ	0.78	0.09	75%	88%	18%	15	7	7
VARIANTE	RSC	1.06	0.09	96%	98%	52%	16	3	3
VARIANTE	SDB	0.80	0.06	68%	85%	24%	12	4	4
VARIANTE	RSC	0.61	0.08	94%	95%	17%	16	8	11
VARIANTE	SDS	0.74	0.05	81%	91%	28%	17	6	10
VARIANTE	BOMZ	0.48	0.10	73%	91%	26%	11	7	9
VARIANTE	RSC	1.28	0.18	95%	97%	36%	20	4	9
VARIANTE	RSF	1.27	0.28	95%	97%	39%	22	3	10
VARIANTE	SDB	0.29	0.03	67%	87%	35%	8	6	6
FNSR	DSTRAT	6.13	0.28	97%	98%	26%	100	8	39
FNSR	RATGRIS	5.08	0.18	96%	97%	48%	90	14	40
FNSR	RSC	2.52	0.67	93%	94%	14%	38	3	15
FNSR	RSF	3.54	1.83	89%	92%	6%	52	6	22
FNSR	SDB	11.70	0.18	95%	97%	74%	173	7	64
FNSR	SDS	2.23	0.50	86%	92%	8%	37	7	17
FNSR	BRECHE	0.67	0.30	65%	66%	19%	435	436	436
FNSR	RATL	0.40	0.25	76%	82%	31%	108	104	105
FNSR	RSC	0.89	1.29	35%	33%	51%	373	372	372
FNSR	RSF	1.39		33%	35%	8%	193	192	193
FNSR	RSC	5.00	0.21	97%	97%	33%	71	4	27
FNSR	SDB	9.47	0.29	97%	97%	44%	147	16	60
FNSR	SDS	2.67	0.96	91%	94%	15%	43	7	19
FNSR	RATL	0.98	0.17	88%	92%	21%	21	7	12
FNSR	RSC	2.99	0.53	95%	96%	28%	51	10	23
FNSR	SDB	7.55	1.50	95%	95%	13%	117	15	50
FNSR	SDS	1.41	0.27	87%	94%	25%	28	9	16
FNSR	DSTRAT	1.37	1.17	91%	94%	31%	106	93	98
FNSR	RSC	2.24	0.49	88%	98%	61%	383	371	375
FNSR	RSF	2.07	1.95	88%	89%	9%	36	7	17
FNSR	SDB	9.63	0.27	93%	97%	58%	144	7	54
FNSR	SDS	2.12	1.35	82%	88%	11%	102	76	85
FNSR	DSTRAT	13.18	0.23	96%	96%	42%	194	3	68
FNSR	RATGRIS	5.06	0.09	95%	95%	26%	76	7	30
FNSR	RATL	0.19	0.06	57%	70%	26%	11	9	10
FNSR	RSC	2.64	0.40	93%	94%	29%	37	4	15
FNSR	RSF	9.00	1.42	95%	96%	18%	127	5	47
FNSR	SDB	10.25	0.17	97%	98%	85%	148	2	52
FNSR	SDS	2.72	0.87	93%	93%	11%	47	10	22
FNSR	SDS	0.23	0.19	40%	67%	23%	288	287	287
OLIVEIRA	BRECHE	2.58	0.30	91%	96%	31%	222	185	197
OLIVEIRA	BR.RAT	1.35	0.49	91%	95%	56%	425	408	414

Fragment	Lithology	Head Grade TCu%	Head Grade TCo%	Leach TCu Recovery	Leach CuOx Recovery	Leach TCo Recovery	TAC	GAC	FAC
OLIVEIRA	DSTRAT	6.28	1.30	92%	95%	45%	107	30	57
OLIVEIRA	RATGRIS	9.91	0.82	97%	98%	26%	148	12	58
OLIVEIRA	RSC	1.26	2.80	74%	75%	4%	27	12	17
OLIVEIRA	RSF	1.69	1.50	88%	90%	11%	30	7	15
OLIVEIRA	SDB	2.27	2.03	86%	88%	8%	43	14	24
OLIVEIRA	BOMZ	3.82	0.24	93%	94%	19%	67	11	30
OLIVEIRA	DSTRAT	4.49	0.83	95%	95%	40%	67	6	27
OLIVEIRA	RATGRIS	4.83	0.79	95%	95%	45%	87	18	42
OLIVEIRA	RSC	1.86	0.55	94%	95%	29%	28	6	14
OLIVEIRA	RSF	3.61	0.35	96%	97%	55%	59	7	25
OLIVEIRA	SDB	5.82	1.66	95%	96%	18%	184	102	130
OLIVEIRA	BRECHE	2.44	1.01	92%	93%	27%	40	10	21
OLIVEIRA	DSTRAT	3.15	1.01	93%	94%	14%	53	9	24
OLIVEIRA	RATGRIS	5.16	0.48	97%	97%	46%	90	13	39
OLIVEIRA	RATL	0.81	0.07	96%	97%	37%	17	5	9
OLIVEIRA	RSC	2.35	0.63	96%	97%	40%	38	8	19
OLIVEIRA	RSF	3.68	0.80	96%	97%	82%	68	16	35
OLIVEIRA	SDB	5.22	1.23	93%	94%	25%	217	142	168
OLIVEIRA	SDS	1.35	0.03	90%	94%	57%	305	287	293
OLIVEIRA	DSTRAT	4.30	0.04	5%	57%	23%	543	542	543
OLIVEIRA	RSC	3.18	0.36	19%	65%	65%	360	358	359
OLIVEIRA	RSF	3.52	0.52	41%	89%	76%	364	360	361
OLIVEIRA	SDB	3.43	0.86	18%	70%	36%	381	379	380
FNSR	BOMZ	4.50	2.48	85%	87%	5%	79	19	40
FNSR	RSC	4.07	0.19	98%	99%	31%	61	4	23
FNSR	RSF	10.12	0.03	97%	99%	82%	140	8	53
FNSR	SDB	10.52	0.07	97%	98%	60%	187	50	96
FNSR	SDS	1.35	0.17	85%	97%	66%	344	331	336
VIRGULE	DSTRAT	10.03	0.03	93%	98%	82%	142	9	54
VIRGULE	RATGRIS	9.41	0.18	96%	97%	19%	149	24	66
VIRGULE	RSC	4.42	0.56	86%	90%	-36%	56	6	23
VIRGULE	RSF	8.46	0.55	96%	96%	14%	127	15	53
VIRGULE	SDB	8.95	0.73	97%	98%	48%	131	4	48
VIRGULE	SDS	3.93	0.28	89%	92%	9%	59	5	23